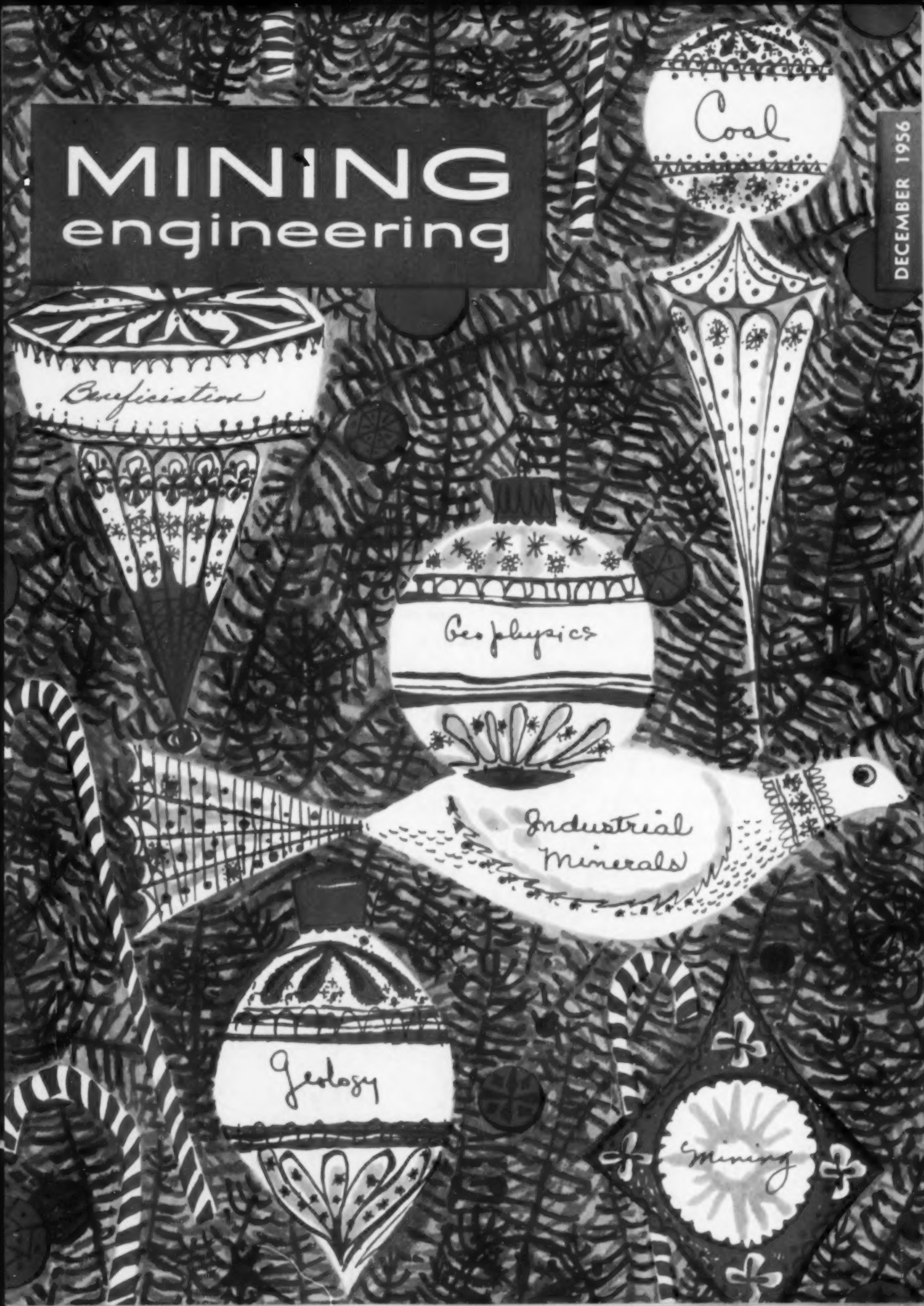
MINING engineering

DECEMBER 1956



MINING engineering

VOL. 8 NO. 12 DECEMBER 1956

COVER

The staff of MINING ENGINEERING and cover artist Herb McClure join in wishing all members of the Mining Branch a Merry Christmas.

FEATURES

LOADING AND TRANSPORTATION SYMPOSIUM

The preceding seven articles are based on a Symposium presented at the Third Rocky Mountain Minerals Conference, Salt Lake City, September 1956.

TRANSACTIONS

The AIME also publishes **Journal of Metals** and **Journal of Petroleum Technology**

Published monthly by the American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., 29 West 39th St., New York 18, N. Y. Telephone: PEnnsylvania 6-9220. Subscription $8 per year for non-AIME members in the U. S., & North, South & Central America; $10 foreign; $6 for AIME members, or $4 additional in combination with a subscription to "Journal of Metals" or "Journal of Petroleum Technology". Single copies $.75; single copies foreign $1.00; special issues $1.50. The AIME is not responsible for any statement made or opinion expressed in its publications.

Registered cable address, AIME, New York. Indexed in Engineering Index, Industrial Arts Index, and by The National Research Bureau. Entered as second-class matter Jan. 18, 1949, at the post office at N. Y., N. Y., under the act of March 3, 1879. Additional entry established in Manchester, N. H. Member, ABC.

PERSONNEL

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc. (Agency) operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Civil and Mining Engineer, 34, graduate C.E., 1943. Experienced open pit and underground coal mining and in design, development, exploration, and construction of mining facilities including preparation plants. Also experienced in negotiation and administration of mineral leases and property sales and purchases including oil, gas, limestone, and coal. Also experienced in railroad construction and improvement of yard and terminal facilities. Will consider locating in North or South America. M-294.

Mine Manager or Mine Superintendent, B.S. in mining, age 46, 16 years varied experience in engineering, development, and production in mines TPD. Working knowledge of Spanish. Prefer western U. S. M-293-San Francisco.

Mining Engineer, 44. Broad varied experience in exploration and development, plant design and construction, production and general overall management principally in open pit methods of operation in U. S. and Latin America. Currently engaged in consulting work on lightweight aggregate production, but available for permanent responsible position. M-291.

Assistant Mine Superintendent, B. S. in mining engineering, 43, eight and one half years graduate engineering and production experience in lead, zinc, and uranium mining, which includes exploration, surveying, mapping, and supervision of mining. Prefer South America. M-292-San Francisco.

POSITIONS OPEN

Mine Superintendent, with supervisory experience for small manganese open pit project. Spanish desirable. Salary, $8,000 to $10,000 a year plus bonus. Location, Mexico. F4214S.

Mining Engineers, to examine salt deposits and to advise on mining and refining. Duration, six months to one year. Location, foreign. F4112.

Geologist, M.S. or Ph.D., under 40, with at least five years experience in nonmetallic fields and economic geology. Must be competent field geologist to handle property evaluation. Salary, open. Location, eastern U. S. W4105.

Assayer-Engineer, qualified to do fire and chemical analyses for common ores and make field examination of mineral prospects. Prefer single status; considerable travel. Salary, to $7000 a year. Position available April 1957. Location, Northwest. W4094S.

Mining Engineer or **Geologist,** to head up an exploration and development program in South America. Should have five to ten years practical engineering experience. Principal interest of company is in barytes ore. Should speak Spanish. Salary, from $10,000 a year, depending upon ability and experience. F3919.

BOOKS

ORDER THE FOLLOWING BOOKS THROUGH AIME— Address Irene K. Sharp, Book Dept. Ten percent discount given whenever possible.

The Origin and Nature of Ore Deposits, by R. T. Walker & W. J. Walker, *The Walker Associates,* \$6.50, 368 pp., 1956.—In an effort to facilitate the discovery and development of ore deposits, this work classifies and describes the various kinds, with emphasis on their chemical and physical characteristics, environment, and mode of formation.

Proceedings of the International Conference on the Peaceful Uses of Atomic Energy, Vol. 6: Geology of Uranium and Thorium, *United Nations Publications,* \$9.00, 825 pp., 1956.—This volume contains 100 papers on the available nuclear raw materials in those countries which participated in the Geneva Conference, August 1955, thereby summarizing an almost worldwide picture of future nuclear energy. The work also includes discussions of prospecting techniques currently being utilized in the search for uranium and thorium.

Clays of Utah County, Utah, by Edmond P. Hyatt, Bulletin No. 55, *University of Utah,* \$2.00, 80 pp., 1956.—This bulletin presents a classification and evaluation of clay deposits in Utah County, complete with maps and catalogues.

Please Order These Publications from the Publishers

Lead Deposits in the Upper Cambrian of Central Texas, by Virgil E. Barnes, *Bureau of Economic Geology,* University of Texas, Austin 12, \$1.00, 68 pp., 1956.—This study, illustrated by scale maps, presents analytical data for deposits of lead and zinc in the Upper Cambrian rocks.

Radioactive Deposits in California, by G. W. Walker, T. G. Lovering, and H. G. Stephens, Special Report 49 of *California Division of Mines,* Ferry Building, San Francisco 11, 52¢, 38 pp., 1956.—A complete descriptive listing of all known occurrences of radioactive minerals in California.

NEMA Standards Publication: Mining Belt Conveyors, *National Electrical Manufacturers Association,* 155 E. 44 St., New York 17, N. Y. \$1.50, 34 pp., 1956.—A compilation of practical information on the construction, nomenclature, and rating of mining belt conveyors.

Third Supplement to Manual on Rock Blasting, K. H. Fraenkel, editor-in-chief, *Atlas Copco Aktiebolag,* Stockholm, available in the U. S. from Atlas Copco Eastern Inc., Paterson, N. J., \$3.00, various pagings, 1956.—Designed to serve as a basic international handbook in the field, the manual provides comprehensive, detailed coverage of rock blasting, and consists of specialized sections contributed by authorities. This supplement contains latest engineering and technical reports on the calculation of charges for both bench blasting and stoping, location and design of permanent compressor plants and a review of tungsten carbide bit development and utilization in percussion drilliing. The material is presented in English, French, German, and Swedish, supplemented by graphs and diagrams.

Geology of the South Manzano Mountains, New Mexico, by J. T. Stark, Bulletin 34, *State Bureau of Mines and Mineral Resources,* Campus Station, Socorro, N. M., \$1.75, 46 pp., 1956.—This study, undertaken to complete the geologic information available on the Manzano Mt. region, is confined essentially to the Pre-cambrian rocks, noting their structural relations to the overlying Pennsylvanian rocks.

The following pamphlets may be obtained from: *State Geological Survey Div.,* University of Illinois, Urbana, Ill., free of charge.

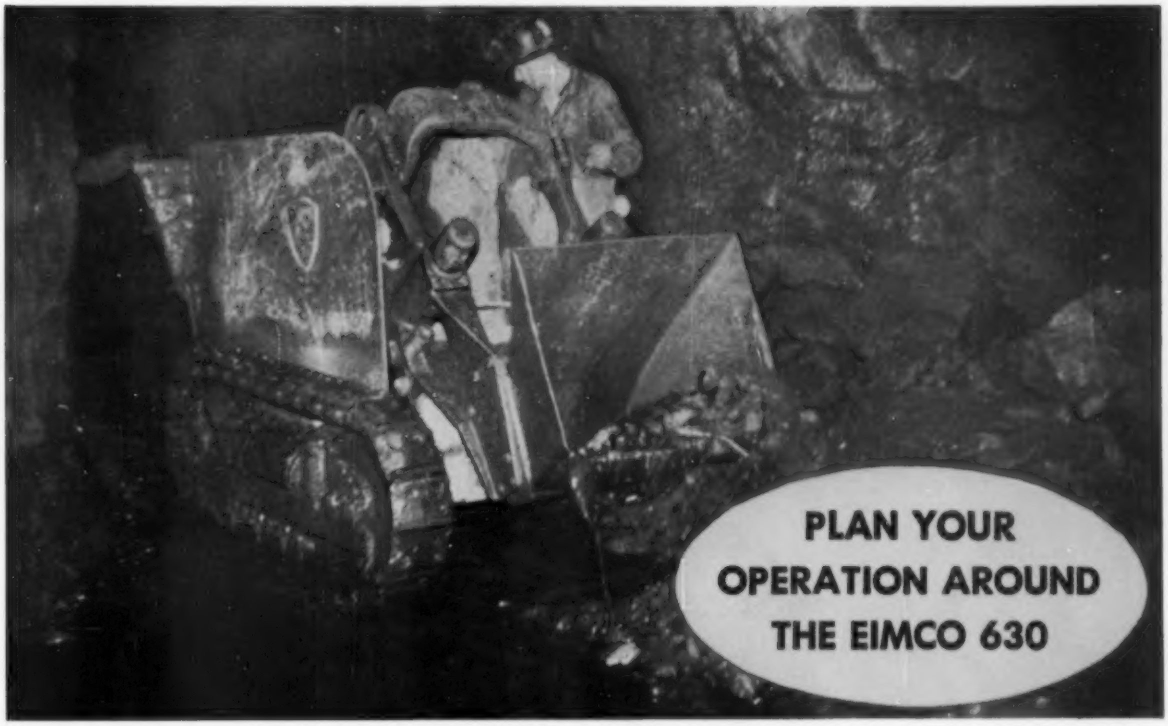
Geologic Structure Map of the Northwestern Illinois Zinc-Lead District, by J. C. Bradbury, R. M. Grogan, and R. J. Cronk, Circular 214, 1956.—The entire mineralized region of northwestern Illinois is here presented in structure maps as an aid to prospecting for zinc-lead ore in the area.

Germanium in Fly Ash and its Spectrochemical Determination, by J. S. Machin and Juanita Witters, Circular 216, 1956.—Report on the germanium content of fly ash in the Chicago area as determined by spectrochemical method, with a detailed description of the method employed.

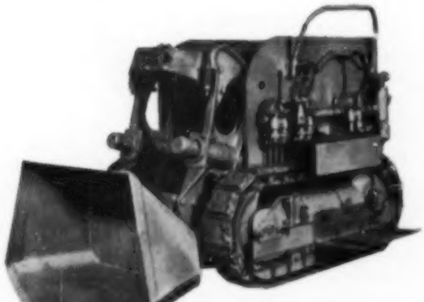
Classification of the Pennsylvanian Rocks of Illinois as of 1956, by Harold R. Wanless, with correlation chart by Raymond Siever, Circular 217, 1956.—A classification based on the cyclical character of the section, giving standard sections for western, central, and southern Illinois in current terminology.

Preliminary Report on Portland Cement Materials in Illinois, by J. E. Lamar, J. S. Machin, W. H. Voskuil, and H. B. Willman, 1956.—A discussion of the economic situation of portland cement manufacture in Illinois, with special reference to manufacturing process, chemical constitution, and nature of the raw materials.

(Continued on page 1150)



Under the most adverse conditions, this Eimco 630 Crawler-Excavator is a production giant. But miners are discovering its capacity is even more phenomenal where mining methods are adapted to the 630's high-production features.



EIMCO 630 EXCAVATOR

EIMCO 630 BULLDOZER

A veteran European mining official—after watching the Eimco 630 Crawler-Excavator load from drawpoints—plans to adopt excavating-loading methods that permit maximum use of the 630's high-production features.

Like any other member of the industry, he wants equipment that gets the highest production rate at lowest maintenance and operating costs.

The ease with which the 630 handles digging and loading chores in confined quarters, convinced this mine manager of its extra-production, economical operating potential before he studied verifying records. And the crew, shift boss, foreman and superintendent were enthusiastic in their appraisal of the machine.

In this mine, the Eimco loads up to 435 tons during a 6½ hour shift where existing conditions do not permit it to load at full capacity. Maintenance costs are less than five cents a ton.

By adopting methods conducive to full capacity operation, production for the newly proposed mine is conservatively placed at 500 tons a shift. Maintenance and operating costs will be one-third that required by competitive equipment used previously.

The Eimco 630 has independent track control for extra maneuverability; easy-to-work levers for operating efficiency; powerful crowding action that quickly fills the big, half-yard bucket; fast discharge ability that cuts loading cycle time, and dependable, rugged construction.

The best way to form an idea how the Eimco 630 will get you more tonnage profit than any other underground mucking machine is to see it in action!

THE EIMCO CORPORATION

Salt Lake City, Utah—U.S.A. • Export Offices: Eimco Bldg., 52 South St., New York City

New York, N. Y. Chicago, Ill. San Francisco, Calif. El Paso, Tex. Birmingham, Ala. Duluth, Minn. Kellogg, Ida. Pittsburgh, Pa. Seattle, Wash. Cleveland, Ohio Houston, Tex. London, England Gateshead, England Paris, France Milan, Italy Johannesburg, South Africa

B-228

Books

(Continued from page 1148)

ORDER THE FOLLOWING BOOKS THROUGH AIME— Address Irene K. Sharp, Book Dept. Ten percent discount given whenever possible.

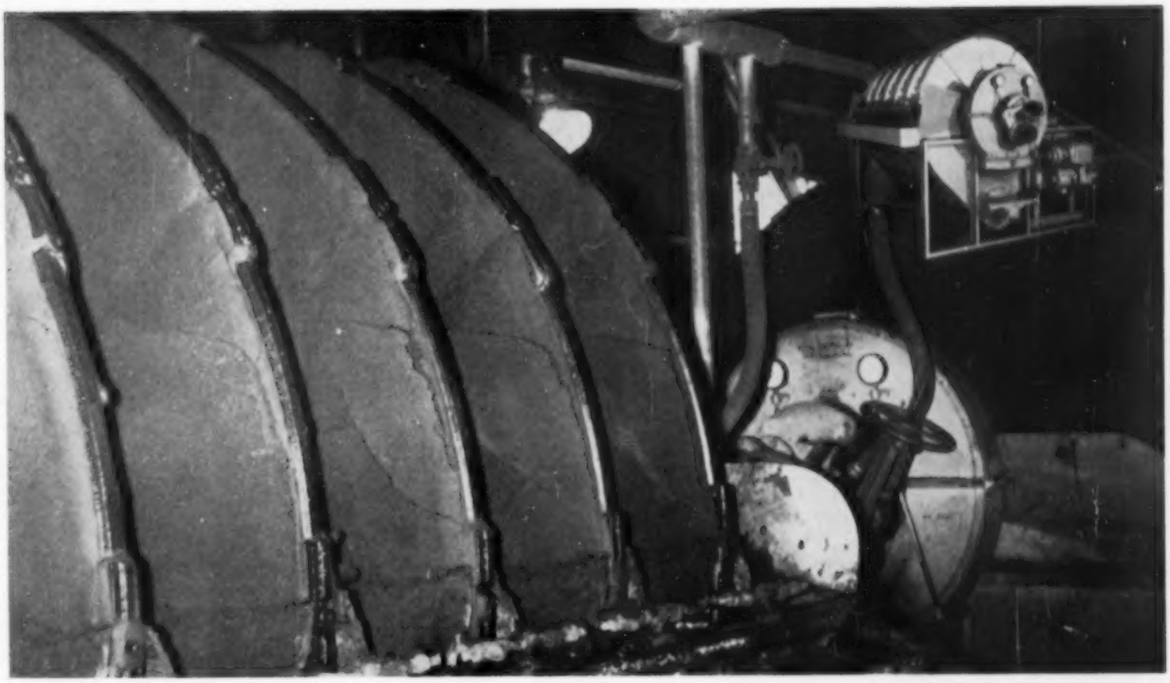
Drainage and Use of Methane from Coal-Fields (Fire-Damp), *Organisation for European Economic Cooperation*, $1.50, 160 pp., 1956.—The report of an intra-European Technical Assistance Mission engaged in the investigation of firedamp drainage in various European coal fields, and the consideration of conditions under which firedamp can be utilized. The Mission was sponsored by the European Productivity Agency, and was composed of experts from Belgium, France/Saar, Germany, Italy, Turkey, the United Kingdom, and the United States.

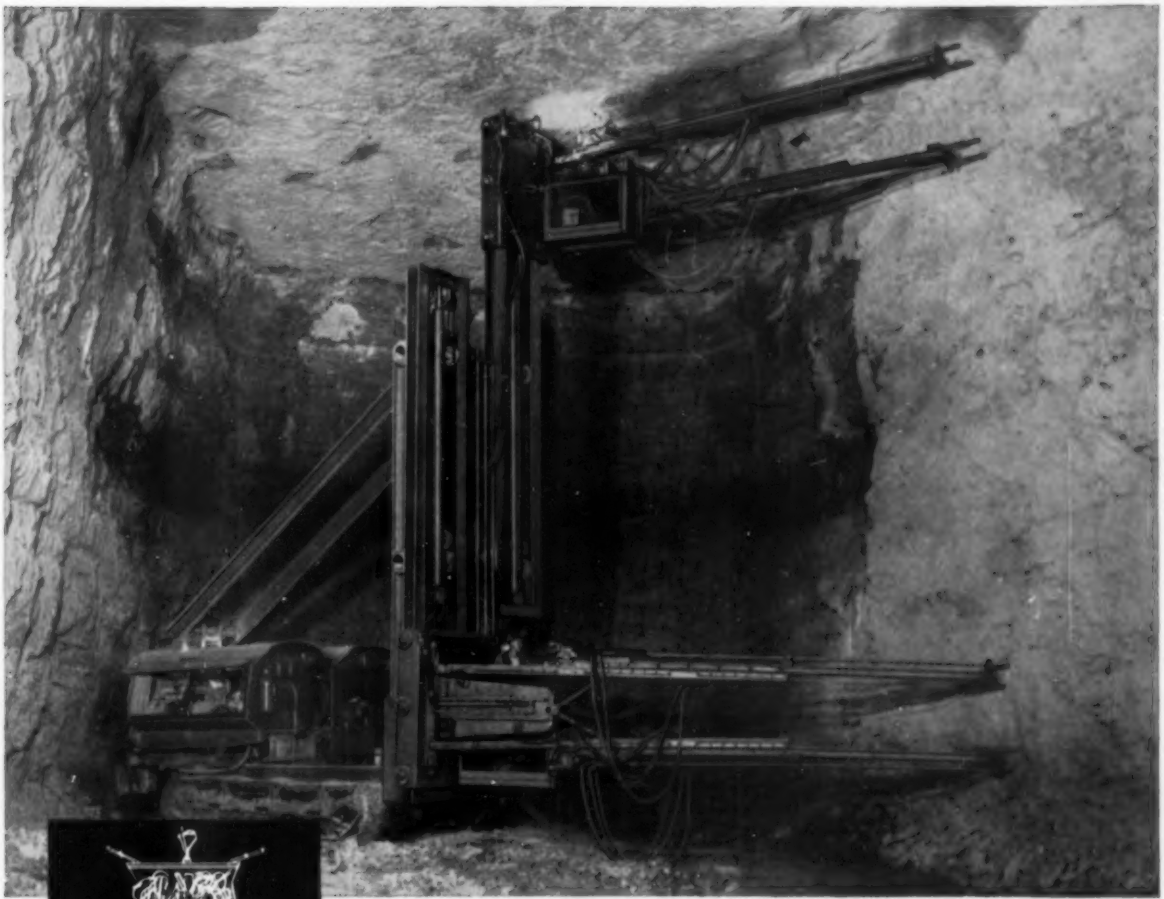
Mineral Resources, Vol. 1: Metalliferous Minerals and Mineral Fuels, by G. A. Kiersch, *University of Arizona Press*, 72 pp., 1956.—This volume describes the location, geologic occurrence, and evaluation of metalliferous mineral and mineral fuel deposits in the Navajo-Hopi Indian reservations of Arizona and Utah. Available from U. S. Bureau of Indian Affairs, Window Rock, Ariz.

Symposium of Papers on Pumping, *Hortors Ltd.*, Johannesburg, Union of South Africa, $3.60, 302 pp., 1955. —These selected papers, reprinted from *The Journal of the Institution of Certificated Engineers*, discuss matters of mine water treatment; pump chamber ventilation; design, testing, and maintenance of centrifugal pumps; electrical drive and control gear; and shaft pump columns.

Symphony of the Earth, by J. H. F. Umbgrove, *Martinus Nijhoff*, $4.00, 215 pp., 1950.—This is the work of a Dutch geologist intended to describe the processes and interrelationships of various topics of earth science in terms interesting and understandable to the layman. Each of the seven chapters has been adapted from a lecture or address and is accompanied by a recommended reading list. The entire text is supported by ten plates and over 100 text figures.

Panel Discussion on Pyrometric Practices, *American Society for Testing Materials*, $1.50, 41 pp., 1956. —This discussion is concerned with ASTM recommended practices on high temperature testing of metals, E 21, E 22, and E 85, with interest focused on defining the precision of temperature measurement and control.





How a new **CAT*** No. 12 can

STEP UP YOUR MINE'S EFFICIENCY 3 WAYS

This new Caterpillar No. 12 Motor Grader maintains 16 miles of haul road and 'dozes truck spillage at the Hill-Trumbull Mine, Marble, Minn. It is owned by the Mesaba-Cliffs Iron Company and operated by the Cleveland-Cliffs Mining Company of Cleveland, Ohio. In building and maintaining haul roads for faster cycle times and reduced wear and tear on equipment, in 'dozing and clean-up work, the new Caterpillar No. 12 does a big and important job. Here is how it can do it at lower cost in *your* mine:

1. LOWER OPERATING COST. The new No. 12 delivers its 115 HP on non-premium, low-cost fuels. Its new oil clutch gives you longer clutch life, easier operation, and as much as 1500 hours between clutch adjustments. Tubeless tires (furnished at no extra cost) run cooler, last longer, and eliminate the tube and flap down time of old-fashioned tires.

2. LONGER WORK LIFE. Like all Caterpillar Motor Graders, the No. 12 is built—not just assembled—by a single manufacturer. This means traditionally sound Caterpillar ruggedness and workmanship, and careful balancing of engine and blade capacity for long life and high efficiency. And it means a single source for parts and service—your reliable Caterpillar Dealer.

3. INCREASED PRODUCTION. Positive, non-creep controls, easy "feel-of-the-road" steering, sure-footed traction with engine positioned over the driving wheels, quick-change blade positioning, unobstructed visibility—all these are good reasons why operators *like* the Caterpillar No. 12 Motor Grader, and do more efficient work on any job.

Your Caterpillar Dealer will demonstrate these and other features of the fast-working, long-lasting No. 12 Motor Grader. See him for proof that the Cat No. 12 will do more work at less cost on *your* job than any other grader.

Caterpillar Tractor Co., Peoria, Illinois, U. S. A.

CATERPILLAR*

99% OF ALL CAT MOTOR GRADERS EVER BUILT ARE STILL AT WORK

Manufacturers News

News
Equipment
Catalogs

• FILL OUT THE CARD FOR MORE INFORMATION •

Wire Screens

Cleveland Wire Cloth & Mfg. Co. employs "engineered weaving" in the manufacture of its complete line of wire screens. State your screen requirements, job application, and specifications, and Cleveland's sample service dept. will send a sample that will fit the job. A new bulletin gives specifications in condensed tables covering popular items. Standard crimps and weaves are available in a wide variety of commercial metals. **Circle No. 1.**

Truck Body

Galion Allsteel Body Co. has a new truck body for rock hauling designed to withstand severe impact shocks. Offered in 6 to 15 cu yd capacities, the units feature ¼-in. steel plate construction in floor, head, and sides. A ¼-in. wear plate over a 2-in. hardwood cushion protects the body floor and a 4x4x½-in. reinforced top roll minimizes loading damage to body sides. A 15° 24-in. scow end eliminates the need for a tailgate. **Circle No. 2.**

Hose Ramps

When flexible lines are thrown across roads or work areas they often make rough riding for vehicles. A new portable hose ramp by *Copperloy Corp.* will provide a smooth roadway and protection when fire

hoses, fuel lines, and air hoses up to 4-in. diam are used where vehicles must pass. Made of lightweight magnesium, the ramps are constructed to give rigid support of 10,000 lb. Diamond safety tread provides good traction. **Circle No. 3.**

Nicad Storage Batteries

Nickel Cadmium Battery Corp. announces a complete new line of batteries containing plates composed of porous sintered nickel impregnated with nickel and cadmium oxides. Long lived, they are claimed valuable for starting off-the-highway vehicles and other heavy equipment and have useful capacity in temperatures ranging from −40°F to 165°F. Cell containers of high impact plastic are grouped in steel cases. **Circle No. 4.**

Self-Propelled Tracdril

Chicago Pneumatic Tool Co.'s G-800 Tracdril is a one-man operated, track-mounted wagon drill. Each track is independently powered by a 6-hp reversible air motor enabling the drill to tow a compressor as well as propel itself about a work area. Motor controls give in-

stant forward, back, or pivot control but return to neutral when released. A 365-cfm compressor will allow full efficiency when a 4-in. drifter or a CP-556 Rotauger is mounted. A track lock prevents the unit from moving during drilling operations. **Circle No. 5.**

Torque Converter Grader

The new Power-Flow model of the Adams 660 motor grader by *Le Tourneau-Westinghouse Co.* has a design similar to the standard 660 but features the operational advantages of a torque converter drive train plus a 27 pct increase in engine power. A 190-hp diesel engine teams with the single-stage converter and a four range constant mesh transmission, to make for

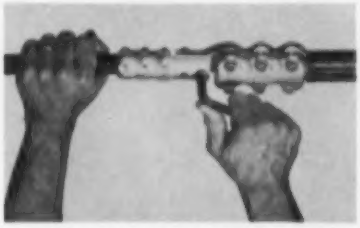
cushioning of shock load, reduction of mechanical wear, and simplified and more efficient operation. Four forward speed ranges vary from .23 to 27.4 mph. Changes in direction are made without hand shifting by the use of a rocker-type pedal that allows the operator to give full attention to steering, blade, and other controls. **Circle No. 6.**

Winch Hoists

Lug-All Co.'s Model 3000-30 ratchet lever hoist is a lightweight 1½-ton capacity unit. Main frame is of aluminum alloy, springs are stainless steel, and the safety handle will bend before permitting a dangerous overload. Free release only operates under no-load conditions and there are no brakes to slip. A cable permits lifting, pulling, or lowering a 1½-ton load a distance of 15 ft; or, using the cable in a single line, a ¾-ton load may be worked a distance of 30 ft. **Circle No. 7.**

Cable Connector

Fast, interchangeable cable connections, which permit adding or removing cable sections in seconds, are possible with the new *Ohio Brass Co.* cap-screw cable connector. Once the halves of the connector have been clamped to the cable ends, their tangs can be overlapped and quickly locked together with two cap screws to form a rugged streamlined connection. Tangs are uniform,



providing complete interchangeability among the five available connector sizes. **Circle No. 8.**

Electric Rail Tractor

A self-propelled rail tractor driven by a storage battery is offered by *Easton Car & Construction Co.* Built to pull two trail cars with a gross load of five tons, the tractor and its trailers operate on 24 in.-gage track at speeds up to 300 fpm. Power is sufficient to assure starting with a full load from a standstill on a curve. **Circle No. 9.**

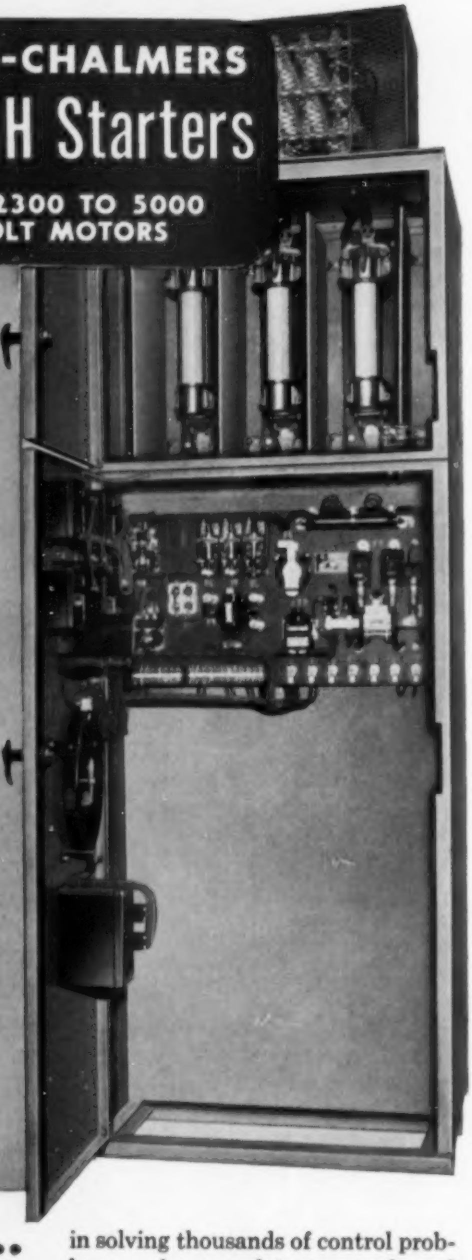
Scraper Hoist

Able to handle scraper loads up to 9900 lb in single line rope pulls, the new *Ingersoll Rand* "41" scraper hoist features rugged construction. In double and triple drum units the barrel-type housing, rope drum, gearing and clutch are arranged in "unit assembly" in which each unit is independent of the next. Some main features include: spring loaded, self-energizing band-type brakes; motors with NEMA standard type D flange mountings; planetary gearing used in primary reduction; nylon insert clutch adjusting nut easily accessible. **Circle No. 10.**

CHOICE of CONTACTORS

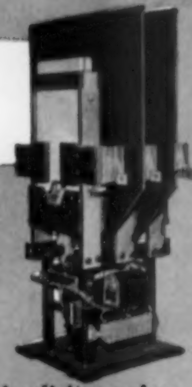
to meet your specific needs

ALLIS-CHALMERS
Type H Starters
FOR 2300 TO 5000 VOLT MOTORS



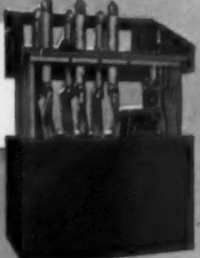
Allis-Chalmers Type H Starters may be equipped with either air-break or oil-immersed contactors —installed in the same sized space.

AIR



Designed for top performance on the rough-tough jobs. Advantage of contacts operating in air include long contact life, reduced fire hazard, easy maintenance. Double-break contacts, vertical action and dual blowouts provide long, dependable operation. Design simplicity makes contactor particularly adaptable for applications requiring frequent starting, inching, reversing, or dynamic braking.

OIL



Meets operating demands of semi-hazardous locations. Contactor operates under oil to prevent sparks from igniting atmosphere and to protect mechanism from corrosion. Contactor is time-proved clapper type. Self-cleaning, rolling-wiping action extends contact life. Self-aligning E-type magnet provides perfect armature seating . . . quiet, maintenance-free operation.

YOU GET MORE...

Allis-Chalmers offers help on specific control application problems. Call your Allis-Chalmers representative. His recommendations are backed by Allis-Chalmers engineering departments . . . by the experience gained in solving thousands of control problems . . . by complete research and testing facilities.

For complete information on the Type H starter, write for Bulletin 14B6410B — Allis-Chalmers, General Products Div., Milwaukee 1, Wis.

ALLIS-CHALMERS

A-4976

(21) DRY GRINDING MILLS: A 44-page catalog of mills for dry grinding and pulverizing is offered by *Hardinge Co. Inc.* Discussed are proper application and selection of conical, tricone, cascade, rod, tube, and disc roll mills for problems in dry grinding. Air classifying arrangements are described, and typical mill applications are shown in a section of pictorial flowsheets. Complete specifications for a number of Hardinge mills are given with detailed performance data on a large variety of materials.

(22) PLUG VALVES: Valve troubles due to restricted port area when handling slurries and viscous fluids may be solved by the use of a round-port lubricated plug valve by *Homestead Valve Mfg. Co.* Valve features a full round opening through its plug and body of the same ID as the pipe it controls. Available in semi-steel, ni-resist, brass, aluminum, and other metals. A 28-page booklet is offered on round-port and other plug valves.

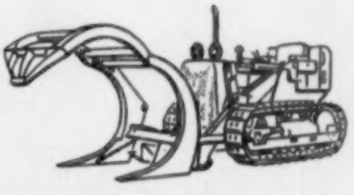
(23) RADIOGRAPHY DATA: "Norelco X-Ray for Industry" is a new 12-page booklet containing operating and application data on six types of radiography units. Specifically intended for industrial testing, the instruments covered include light portable self-contained types and large units of 150 and 300 kv. Information is given also on an industrial image intensifier and closed-circuit television installations in this booklet from the Instruments Div., *North American Philips Co. Inc.*

(24) WHEEL TRACTOR & SCRAPER GUIDE: A comprehensive operator's guide for wheel tractors and scrapers is offered by *Caterpillar Tractor Co.* Form 32022 stresses operator benefits—easier operation and better performance. Detailed tips cover loading, hauling, dumping, and returning. Basic service checks are made simple with the use of a checklist of important pre-operation steps.

Free Literature

(25) BLASTING MACHINES: A 28-page manual from the Explosives Dept. of *Hercules Powder Co. Inc.* includes a catalog of the Hercules line of blasters. Sections on testing and care are included with wire connection methods and safety precautions. In addition, a list of "don'ts" supplements a section of general remarks. Hercules blasting machines detailed include the 50-cap, 30-cap, 10-cap, and 10-cap permissible.

(26) SKID-SHOVELS: A new catalog of diagrams and on-the-job photos of the *International Harvester Co.* line of Drott Skid-Shovels lists



latest improvements. Break-out action, hydro-spring, four-in-one features are some advantages designed to give large capacity, low upkeep, and long life for these machines. Attachments are also detailed in this 16-page brochure, K-656.

(27) TRACTOR-COMPRESSOR: *Le-Roi Div.* of Westinghouse Air Brake Co. has a combination 125-cfm air compressor and 42-hp wheel type tractor designated the Le Roi 125 Tractair. Available are a number of case histories of the unit at work with several attachments that extend its versatility.

(28) FLEXIBLE CHAIN COUPLINGS: *Morse Chain Co.* has a revised chain coupling catalog that also tabulates data on stamped steel covers, plasticones, and split aluminum covers. Specifications, dimensions, ratings, and applications are detailed on series DSC and SA silent chain couplings, and on series DRC roller chain couplings. General and ordering information is included.

(29) CENTRIFUGAL PUMPS: A new 4-page folder of centrifugal pumps for handling abrasives, corrosives, and acids is available from the *Allen-Sherman-Hoff Pump Co.* Interchangeable rubber-lined slurry pumps offer a choice between the Hydroseal's protective flow of sealing water, and the Centriseal's ability to deliver abrasive or corrosive pulps undiluted. Both types are identical except for two replaceable wearing parts. Hydroseal sand, dredge, and vertical sump pumps are also described.

(30) VIBRATING SCREENS: Construction features of *Allis-Chalmers* vibrating screens for handling feed up to 6 and 20-in., and coal up to 8 and 24-in. are described in a new 24-page bulletin. Both screens feature a patented, balanced, 2-bearing mechanism and circle throw action which imparts uniform vibration to the entire screen surface. Model SH is designed for a wide range of materials and is available in single or multiple deck models. Extra heavy duty Model XH is used primarily in scalping following large jaw or gyratory crushers and is available with plate decks for scalping slabby materials or with rod decks for coarse scalping of sticky materials.

(31) LAB SERVICES: Consulting, inspecting, and testing facilities of *Abbot A. Hanks Inc.* are described in a 90th Anniversary brochure. Laboratory services include chemical analysis, assaying, metallurgical supervision. Engineering investigations include laboratory and field inspection work.

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

12

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

Not good after Mar. 15, 1957—if mailed in U. S. or Canada

Please send { More Information ☐ / Price Data ☐ / Free Literature ☐ } on items circled.

Name__________ Title______

Company__________

Street__________

City and Zone__________ State______

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
21	22	23	24	25	26	27	28	29	30
31	32	33	34	35	36	37	38	39	40
41	42	43	44	45	46	47	48	49	50
51	52	53	54	55	56	57	58	59	60
61	62	63	64						

Students should write direct to manufacturer.

(32) REAGENT FEEDERS: For accurate feeding of frothers, solutions, and slurries the *Clarkson Co.* has the Model E feeders in corrosion-resistant stainless steel. Handling amounts from drops per min up to 2400 gpd, the feeders are claimed efficient and economical. Stainless construction adds to appearance and low maintenance cost. For acids and acidic solutions the Model E in PVC is available.

(33) MAGNETIC SEPARATORS: Bulletin 846 from *Jeffrey Mfg. Co.* details features of drum type separators for use in magnetic recovery and wet concentration problems. Electro-magnets housed within the drum are stationary and supported on a hollow shaft on which the drum rotates. They are adjustable to varying requirements and require no slip rings or contact brushes. Described in detail are the type C magnetic cobber, Jeffrey-Steffensen counter-flow separator, and type CO and CS separators.

(34) ACID HANDLING PUMP: *Dorr-Oliver Inc.* announces a new bulletin describing the Olivite acid handling pump which combines simplicity and efficiency with corrosion resistance. Casting and cover are lined with Olivite, a permanently bonded rubber base composition of high tensile strength. Pump capacity ranges from 5 to 1400 gpm with heads up to 120 ft. Data includes equipment photos, cross-sectional drawings, and performance and power requirement graphs.

(35) GRINDING—WET & DRY: Rod, ball, pebble, tube, and compartment type mills with grate, overflow, or peripheral discharge are described in 12-page Bulletin 232 from *Nordberg Mfg. Co.* Complete line of mills detailed range from 8½ to 13 ft diam and up to 50 ft long. Typical installations are shown; and also included are photos of Nordberg's facilities for mill casting, fabricating, machining, assembly, shipment.

(36) TRAXCAVATORS: "New Traxcavators for New Profits" is an 8-page booklet from *Caterpillar Tractor Co.* covering three units—No. 933, No. 955, No. 977. Such performance factors as stability, bucket capacity, power, and speed are pointed up in relation to their effect on profit. Production figures on specific jobs accompany photos of these Traxcavators at work.

(37) AIR POLLUTION CONTROL: A 4-page article, "Tools and Techniques in Air Pollution Control," is offered by an air pollution consultant firm. Staff-written, the paper from *Hemeon Assoc.* includes discussions of new-plant surveys, dust collectors and fly ash arrestors, stack emission surveys, stack gas tracer studies, community surveys for ordinance and zoning development.

(38) CIRCUIT BREAKERS: *Heinemann Electric Co.* has a revised bulletin covering general purpose circuit breakers of the hydraulic-magnetic type. Engineering data on basic design, voltage drop curves, interrupting capacities is included with schematic diagrams and other information to aid both plant and design engineers.

(39) PNEUMATIC PROCESS CONTROLS: A 16-page catalog of pneumatic instruments for process control has just been released by *United States Gauge.* Discussed are indicating pilots, transmitters, and receiver gages. Complete information is contained on measuring elements for pressure and temperature applications.

(40) UNPLASTICIZED PVC PIPE: Data on the Van-Cor line of unplasticized polyvinyl chloride pipe, fittings, and valves is given in a 16-page engineering catalog from *Colonial Plastics Mfg. Co.* Contents include physical properties; chemical resistance tables; pipe, fitting, and valve dimensions; flow rate charts; and assembly and installation pointers.

(41) LITERATURE INDEX: *Minneapolis-Honeywell Regulator Co.* has a new index of product, technological, and application literature. Bulletin G-2 (superseding Bulletin 100-D) covers all catalogs, bulletins, specification sheets, illustrated lectures, and articles from the company magazine *Instrumentation.*

(42) CENTRIFUGAL FANS: A new design in backward curve airfoil centrifugal fans is announced in a 64-page illustrated booklet by *Chicago Blower Corp.* Smooth even flow of air over the blade is claimed to eliminate eddy currents, reduce noise, increase efficiency, and reduce required power. Self-cleaning, the fans are made in five classes. Diameters range from 13½ to 168 in.

(43) INDUSTRIAL FURNACES: Bulletin SC-175 from *Surface Combustion Corp.* describes 27 models of Surface Standard heat treat furnaces. Units covered range from small laboratory furnaces to continuous snap hearth, chain belt, and brazing furnaces. Also available from Surface is special equipment in a line from pilot units to automatic heat treat plants.

(44) POCKET pH METER: Scientific Instruments Div., *Beckman Instruments Inc.* offers data on a new battery-operated pocket-size pH meter. Meter has a range of 2 to 12 pH, and reference and glass electrodes are combined in a single dip rod. Useful for laboratory work and in spot-checking pH equipment, the instrument also has a memory dial for standardizing.

(45) WHEEL BLOCK: Operators of off-the-highway heavy vehicles will be interested in a ruggedly constructed cast steel wheel block by *Calumet Steel Castings Corp.* It is designed to chock tires up to 60-in. diam and loads in excess of 50 tons without failure. Dimensions are 10x10x12-in. and weight is approximately 40 lb.

(46) GROMMET V-BELTS: Construction features of Texrope grommet V-belts, said to provide 20 to 50 pct longer life than other belts, are detailed in a new bulletin released by *Allis-Chalmers Mfg. Co.* Other advantages claimed over ordinary belts include less shrinkage, less stretch, cooler running, and better gripping. As no splice is used, the belts are uniformly strong.

(47) PILOT PLANT FURNACES: A 6-unit line of furnaces made specifically for pilot plant operation is offered by *Lindberg Engineering Co.* Available are: atmosphere box furnace for continuous operation at 2600°F, box furnace with reactor control, 3 kw high frequency induction furnace, horizontal tube furnace, vertical tube furnace, pot furnace for operation to 2500°F.

FIRST CLASS
PERMIT No. 6433
Sec. 34.9 P.L.&R.
New York, N. Y.

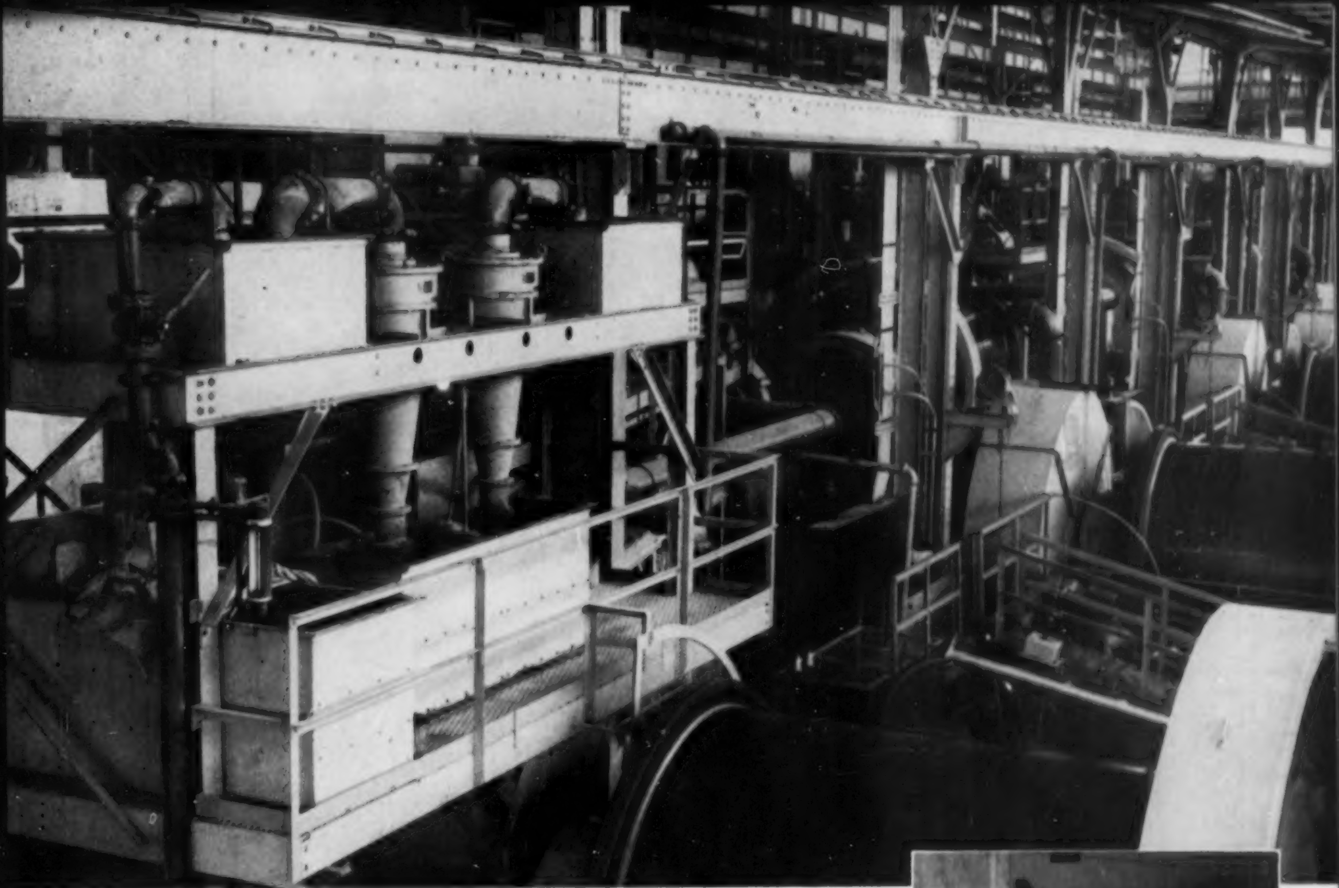
BUSINESS REPLY CARD
NO POSTAGE STAMP NECESSARY IF MAILED IN THE UNITED STATES

3c.—POSTAGE WILL BE PAID BY—

MINING ENGINEERING

29 WEST 39th STREET

NEW YORK 18, N. Y.



Two model D20 Krebs Cyclones in closed circuit with 9' x 11' ball mill

Two model D20B Krebs Cyclones on the sand-slime separation

Krebs Cyclones in closed circuit and for sand-slime separation

Homestake Mining Company uses Krebs Cyclones for closed circuit classification with their new grinding unit and for the sand-slime separation. The cyclones, specially designed for each objective, are low in cost and easy to operate. The new classification techniques give sharper cuts and better percolation rates. Hundreds of model D20, D20B and EE20-9B cyclones are in service and all are operating with the original molded replaceable body liners and long sweep inlet nozzles. Many of these have now handled over a million tons of ore per cyclone.

EE PRODUCTS

EQUIPMENT ENGINEERS INC.

41 SUTTER STREET SAN FRANCISCO 4, CALIFORNIA

Manufacturers of Krebs Cyclones, Valves and Clarkson Feeders

Great Lakes Gypsum Expansion

National Gypsum Co. has begun a $19 million Great Lakes area expansion program involving the construction of two gypsum building products plants and the development of a 75-million ton northern Michigan gypsum deposit. The deposit is located under 48 ft of overburden on a recently purchased 2700-acre tract near Tawas City.

Vitro to Expand Facilities, Extends AEC Contract

Vitro Uranium Corp., Salt Lake City and the Atomic Energy Commission have signed a contract providing for an increase in plant capacity, installation of a new process, and extension of an AEC agreement to purchase uranium concentrates. Continuance of the Salt Lake City operation is provided through March 1962. Vitro will install a new and more efficient extraction method, the liquid-liquid solvent extraction process. Changeover and expansion is expected to cost a minimum $1,200,000.

Armco Plans Opening of New Coal Mine

Armco Steel Corp. has announced application for a certificate of necessity covering a $5 million project to open a new coal mine near McAlester, Okla. Reserves of coking coal in the area are expected to be adequate to fill the needs of the Houston Works of the firm's Sheffield Div. where a $118 million expansion program has been proposed.

Rutile From Black Sands of Australia

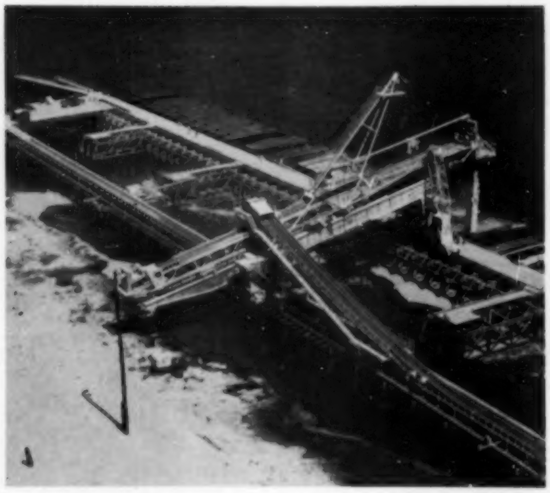
Australia is yielding large quantities of the titanium ore, rutile, derived from the black sands found just below the surface of its beaches. Last year Australia supplied more than 50,000 tons, representing approximately 15,000 tons of titanium. The other two major sources, Mexico and the U. S., produced 24,000 and 10,000 tons respectively.

Alabama Magnesium Plant Projected

A $7 million plant for the annual production of 10,000 tons of high purity magnesium has been planned by a new corporation at Selma, Ala. Alabama Metallurgical Corp., jointly owned by Dominion Magnesium Ltd. and Brooks & Perkins Inc., plans to reduce magnesium oxide, obtained from dolomite, with ferrosilicon. Options have been taken on 480 acres on the Alabama River and the plant is expected to begin operation in 1957. Use will be made of a dolomite quarry at Montevallo, Ala., about 75 miles from the proposed plant site.

Prototype Plant to Produce Manganese

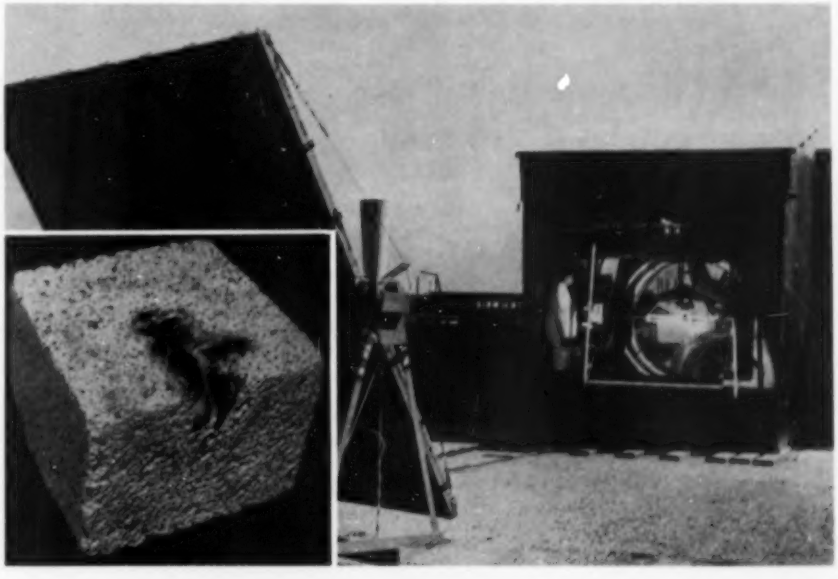
A subsidiary of Strategic Materials Corp., Strategic-Udy Metallurgical and Chemical Processes Ltd., has placed in operation an electric furnace plant at Niagara Falls, Ont. Initial operation will be the processing of some 5000 tons of manganese ore from New Brunswick where another subsidiary, Stratmat Ltd., holds an estimated 150 million tons of reserves. Primary work is intended to provide data which will be applied in the planning of a commercial plant proposed to initially produce 75,000 tons of ferromanganese annually.



New shipping facilities of Freeport Sulphur Co. at Port Sulphur, La., were designed to halve the average loading time for Mississippi ships and barges. ABOVE: Traveling loader can move 400 ft along dock and its discharge chute will extend 45 ft over the river and clear an elevation up to 54 ft. RIGHT: Beyond the traveling loader a stationary tipple prepares to load an ocean-going vessel.

New research laboratories of National Carbon Co., div. of Union Carbide & Carbon Corp., at Parma, Ohio. Basic study here will emphasize solid state physics, of which the best known product to date is the transistor. Located near the center of a 126-acre plot, the laboratories contain approximately 175,000 sq ft of floor space.

Solar furnace installation atop Kennecott Research Building on campus of University of Utah, Salt Lake City. Plate glass mirror in foreground catches sun's rays, directs them to a tilted mirror in background, which transfers them on to a dome-shaped mirror above. This parabola concentrates the light downward to a focal point, where heat attained has exceeded 5000°F. Research materials may be kept in glass containers to control conditions of atmospheric purity. Note burned area of fire brick (inset) after few seconds under heat.



"Thiokol" synthetic rubber, is an organic polysulfide elastomer. One of its many uses is in solid propellents for long range and high altitude missiles. In liquid form, "Thiokol" synthetic rubber mixed with an oxidizer, is poured into specially designed combustion chambers of rockets. It helps to give stability to the fuel charge and resistance to shock. It promotes uniform burning. When the rocket motor is ignited the mixture burns with great intensity and generates large volumes of gas to propel the rocket.

Solid propellents made with "Thiokol" synthetic rubber have proved their value in rockets over liquid propellents in many ways: they are less costly and easier to manufacture—simple and rugged construction makes handling and launching easier and safer—fuel tanks and complicated feed systems are eliminated.

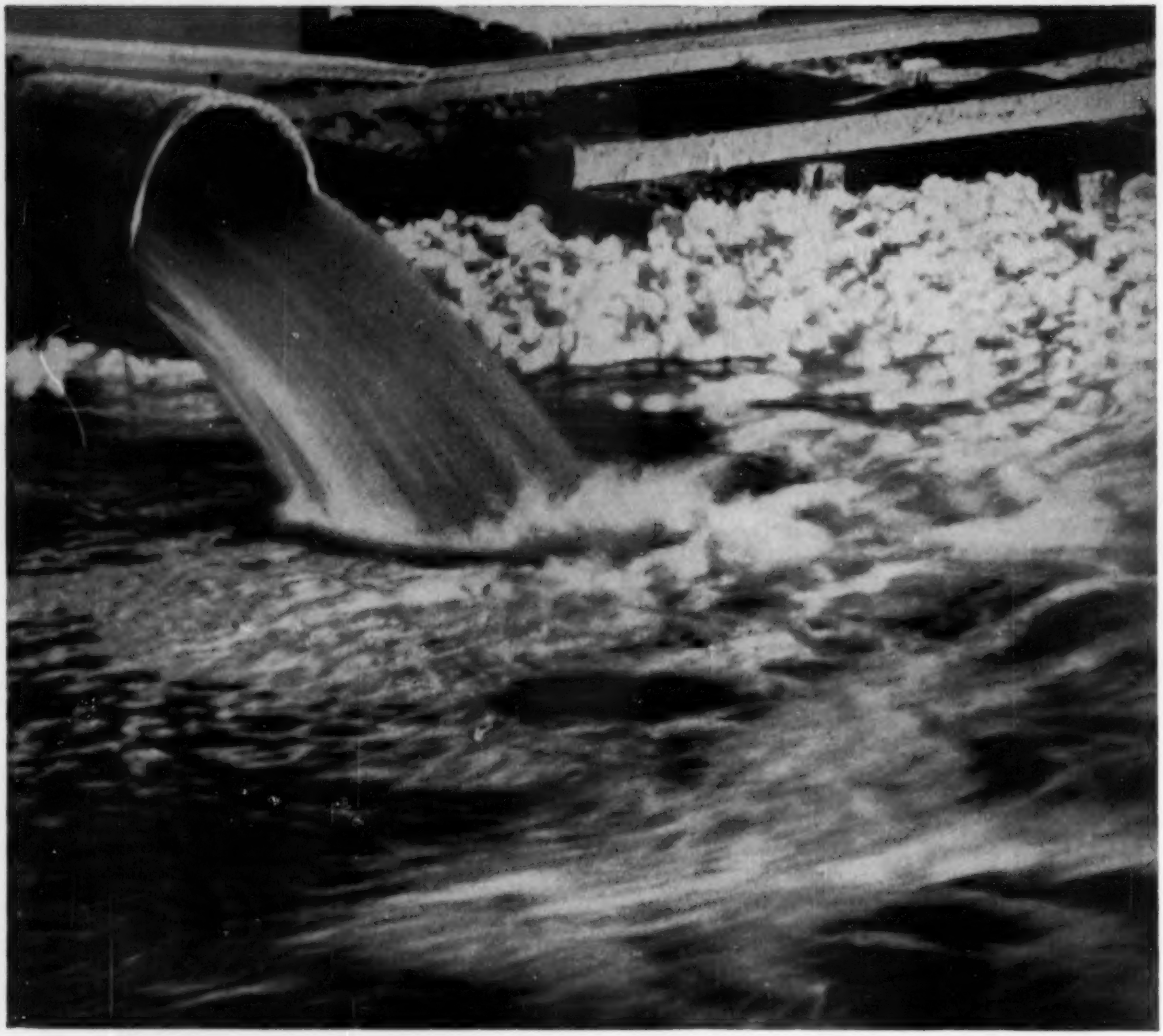
"Thiokol" synthetic rubber is a product containing a high percentage of Sulphur—its name being derived from the Greek words for sulphur and glue. Here is another example of the continually broadening field in which Sulphur is an important and necessary element.

**A trade name of Thiokol Chemical Corporation.*

Texas Gulf Sulphur Co.

75 East 45th Street, New York 17, N. Y.
811 Rusk Avenue, Houston 2, Texas

Sulphur Producing Units
- Newgulf, Texas
- Moss Bluff, Texas
- Spindletop, Texas
- Worland, Wyoming



AMC Mining Show Draws Large Audience

Mining men, over 7000 strong, congregated at Shrine Exposition Hall, Los Angeles for the 1956 Mining Show of the American Mining Congress. A well-rounded program of technical and economic problem sessions, and tours of the exhibits of many major mining firms filled the meeting days, Oct. 1-4. Field trips were programmed for Oct. 5.

Convention sessions covered every phase of mining from national mineral policies to the every-day operating problems of mines, quarries, and processing plants:

Monday, Oct. 1—Opening of the first session, National Mineral Policies at which Secretary of Interior Fred A. Seaton pledged submission to the next session of Congress of a long-range minerals program designed to aid the full development of U. S. mineral resources. Following was a session on Labor Relations and a technical discussion, Milling and Metallurgy.

Tuesday, Oct. 2—Sessions were held on: Developments in Industrial Minerals, Exploration and Geology, Uranium Milling, Tax Panel (Gold, Silver, and Monetary Policy), Drilling Symposium, Open Pit Mining. These were followed by an evening Miner's Jamboree at the Hollywood Palladium.

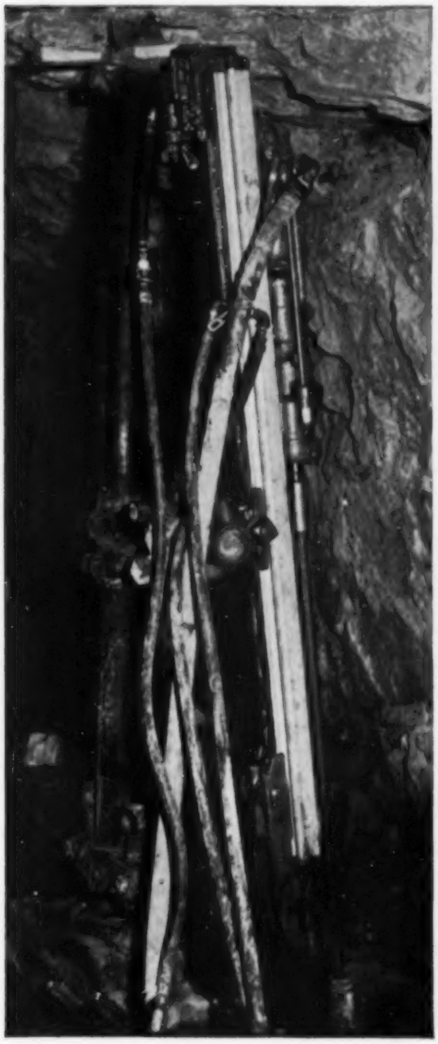
Wednesday, Oct. 3—Meetings covered: Public Land Problems, Underground Mining, Milling and Metallurgy, Management Problems, Industrial Minerals—Production and Processing, Uranium Exploration and Mining.

Thursday, Oct. 4—The last day of major sessions covered: State of the Metal Mining Industries, Open Pit Mining, Health and Safety Conference, Uranium in the Future, Underground Mining, Strategic Minerals Conference.

During these session days, an exposition of the newest in machines, equipment, and supplies for mining, quarrying, and mineral treatment was held in the Shrine Exposition Hall and representatives of manufacturers eagerly helped those with problems and gave enthusiastic explanations of the applications of their products.

The Annual Banquet was the climax of the show and top performers entertained a capacity audience at the Embassy Room and Cocoanut Grove of the Ambassador Hotel.

Gardner-Denver . . . Serving the World's Basic Industries



Reduce ore breakage costs by 50% to 75% with Gardner-Denver Deep Hole Drills

Permit ring drilling to 100 feet or more from a 7-foot development drift. Also for bench holes, stoping, slashing, pillar recovery.

Engineered deep hole equipment includes 4" or 4½" drills, long feed aluminum guideshells, Ring Seal Shank, sectional rods and couplings, bit adapter for carbide bits.

Send for illustrated bulletin on deep hole drilling . . . it's packed with application reports and equipment specifications.

GARDNER - DENVER

THE QUALITY LEADER IN COMPRESSORS, PUMPS, ROCK DRILLS AND AIR TOOLS FOR CONSTRUCTION, MINING, PETROLEUM AND GENERAL INDUSTRY

Gardner-Denver Company, Quincy, Illinois

In Canada: Gardner-Denver Company (Canada), Ltd., 14 Curity Avenue, Toronto 16, Ontario

A VISITOR to a uranium ore treatment plant expressed the idea that, "basically this is just a cyanide plant." Some justice in that—and a good chance to remind ourselves of the heritage that MBD'ers (and chemical engineers) have received from the cyanide plant operators of the late 1890's and early 1900's. Practically all the present unit operations were born then, or the necessary equipment devised to make the operations continuous.

There was intensive development of cyanide plant flowsheet and equipment starting at the turn of the century and continuing for a decade or two. As the period came to a close there were available efficient machines for grinding fine, classifying, sizing, moving pulp, agitating, getting solution in, getting it out, and for the dewatering steps—thickening, filtering, drying.

Today's nonferrous hydrometallurgist has made his contribution with the newest unit operation: ion exchange. However, one finds this new operation being fitted into a plant flow pattern that had largely crystallized 40 years or more ago. This is not an indication that new plants do not contain many innovations, but rather, good evidence that the basic ideas developed half a century ago are still sound.

◆◆◆◆◆◆

OVER 200 million net tons of raw steel has been produced since the end of World War II in the Eastern steelmaking district—with nearly two-thirds of its capacity located not more than 100 miles from Philadelphia, said Max D. Howell, executive vice president of American Iron & Steel Institute, in Philadelphia recently.

"Even more spectacular has been the expansion of steelmaking capacity in this district. The potential output of steel in this district increased over 9.5 million tons from 1946 to the start of 1956. No other district can match that gain. It was a 53 pct increase, compared to a national average increase of about 40 pct." Over 17 million tons of steel can be made within 100 miles of Philadelphia, and more than 27 million in the entire Eastern district, he said.

"Great as its progress has been, this thriving region is on the verge of additional dynamic growth," said Mr. Howell. "Most of the steel plants in the 100-mile semicircle around Philadelphia are expanding rapidly. Simultaneously, this area is becoming increasingly important for the amount of foreign iron ore being handled here. You are fortunate to have good water and rail transportation, good markets, good plant sites and other advantages."

The amount of steel in use in the U. S. is increasing sharply as this country's flourishing economy continues, said Mr. Howell. He listed numerous specific markets that offer good prospects for the steel industry, in addition to the increased amounts of steel that will be needed for the growing population.

Highway construction will require nearly 49 million tons of finished steel in 13 years, plus large additional tonnages for road building machinery, he said. Another important prospect is pipeline construction, which may boom because of the Suez Canal crisis. The expanded shipbuilding program, the St. Lawrence Seaway, power plant construction and a more active freight car program, are still other prospects.

"The American economy has been built on a foundation of steel," said Mr. Howell. "We have completed the first century of the steel age, and during that time we have made more than 3 billion tons of steel in this country."

◆◆◆◆◆◆

IT has been said that operations research is a name for the application of more scientific methods to the solution of management problems. It may be likened to the addition of new scientific tools to a known and proven useful kit. But, to an operating executive, involved in constant day-by-day and long-term decision making, understanding and acceptance of an advanced scientific concept such as operations research becomes in itself a monumental task. This is increasingly so because of the newness of the tool and the difficulty of its proponents in reducing it definitely to terms which will carry meaning to the uninitiated.

At a recent conference of the American Management Association, M. I. Dunn, vice president-operations for the Chesapeake & Ohio Ry. gave examples of the application of this new tool to his company's problems. He went on to express these views on the relationship of the administrator to the use of what he referred to as "O.R."

"Speaking for myself as an administrator, I must admit that the facets of Operations Research which involve application of advanced mathematical procedures in arriving at answers to O. R. problems must be adopted as valid on my part to the extent that I have faith in the integrity and full competence of those people who produce the solutions.

"This implies that for a given problem in the field of operations, a preliminary study should normally be made in order to arrive at a decision as to whether O. R. tools should be used, or whether less advanced methods may prove adequate.

"If the factors involved can be reasonably evaluated with less advanced processes, it is questionable whether O. R. as such need be resorted to.

Herbert Solow, in *Fortune*, puts it this way:

The O. R. worker must balance the costs of study against the economies a solution may effect, and the probability of reaching a solution. He must build no mathematical mountain to produce mouse-size conclusions.

"When problems of sufficient magnitude, involving many or conflicting ramifications which cannot be resolved by common methods and procedures, face us, the advisability of search for more scientific methods of solution is indicated.

* * *

"I begin increasingly to see that nearly all of the problems of expansion we have solved in the past have resulted in solutions which often failed to evaluate some of the less apparent factors and duly influence our final decisions. The resulting physical plant expansion and methods of operating, hence, often failed to generate optimum benefits for the company and the railroad industry as a whole.

"Management needs to have the whole picture before making decisions of this scope. Management needs help in getting the whole picture.

"So, with all the emphasis we find today on mechanization and automation, we find ourselves in the final analysis coming back to the importance of people—highly trained, dedicated people—people with a highly-developed, insatiable curiosity as to Why? (or Why Not), How? When? Where? What? People, in short, to whom the scientific approach is a way of life."

◆◆◆◆◆◆

THE steel industry—searching in every direction for a domestic source of the manganese it must have for its furnaces—has sponsored research on three promising approaches. At a Philadelphia regional meeting of American Iron & Steel Institute, B. P. Schofield, a chemical engineer at Armour Research Foundation of Illinois Institute of Technology, described work with two chemical processes under the sponsorship of AISI. He explained that both processes originated at steel companies, and were passed on to Armour for development.

"Manganese is available internally to this country," Mr. Schofield said. "The problem lies in recovering it in a form usable to the steel industry. It is available as deposits of low grade ore varying in extent, composition and location, and also as a constituent of open hearth slags. In the latter form, it is readily available to each steel producing center and is as concentrated a source of manganese as the average low grade ore, since both are in the general range of 10 pct."

The author explained that the two chemical processes described in his report used chlorides of hydrogen and calcium, respectively, as agents in the recovery of manganese from open hearth slag. Summarizing the findings, to date, Mr. Schofield declared that chloridization with hydrogen chloride appears "quite feasible" and that he sees no basic difficulties to warrant the elimination of the calcium chloride process.

◆◆◆◆◆◆

AEC releases in recent weeks have made possible a rough tabulation of the status of nuclear power reactor development in the U. S. Two conclusions may be drawn at the moment: 1) that the U. S. is temporarily lagging in actual power production in comparison with Britain, France, and the USSR; 2) that completion of any one of the projects now under construction will put the U. S. out in front by a wide margin.

Apparently the story is that U. S. power producers, with the world's richest fuel resources on tap, have felt less pressure to prematurely rush into commercial nuclear power production. On the other hand, they have not lagged in research or design and this program will bear bigger fruit when it matures.

At present the AEC lists three "full scale civilian power reactors" under construction, with 11 more planned in this category, and several "proposed" to the Commission. U. S. manufacturers are listed as participating in six projects for civilian power reactors in locations in Europe, Latin America, and the Philippines.

Including experimental, research, and military power reactors the tabulation on the program to date is as follows:

A. Low Temperature (Not useful for power generation)

Completed, Later Dismantled	9
Completed and in Being	59
Under Construction	27
Planned	36
Total	131

B. High Temperature Power Producing Reactors

Completed, Later Dismantled	5
Completed and in Being	9
Under Construction	22
Planned	47
Total	83

This brings the grand total of reactors for various purposes, military, research, and power, to 214. The largest of the reactors announced to date is that of the Consolidated Edison Co. of New York. Projected to cost about $55 million, the reactor is planned to yield 236,000 kw.

MINING engineering DRIFT

Affiliate Membership in EJC Evaluated

THIS is in answer to many requests for information concerning the role of Engineers Joint Council in the engineering profession, particularly the role of its Affiliate Member societies.

The profession is represented by societies formed of individual members having a common technical interest. To facilitate communication between members, local sections or chapters of national societies have been formed to operate in communities or geographical areas. In addition, there are many local or regional societies composed of members from the several branches of engineering.

The technical phase of the profession has been developed to a high degree by each society, but the impact of the profession on the public by the individual, his local section, or national society requires a coordinated effort by all the segments of the profession. Engineers Joint Council provides the medium for this effort.

Associate and Affiliate Societies

Associate Member societies are national in the activity and technical interest of their members, differing only in size from the larger Constituent societies.

The Affiliate Member societies are composed of either of two types of organizations, or even a combination of both. One is the local or independent regional society, organized and operating for the benefit of the engineers in an area. The other, of greater importance here, is the federation of engineering societies, or local sections or chapters of such societies.

Since all individuals trained in the engineering profession are engineers first and specialists second, there has developed the need for a common medium of expression. The federation of local sections and/or regional societies provides a forum for specialists with the one common interest, the profession of engineering, to become associated in affairs of public interest.

Engineers, as professional persons, are obligated to devote part of their time and effort to matters affecting public interest. In order to help the engineer obtain proper public recognition and professional status, the public must know of him both as an individual and as a member of a group. The federation provides the medium through which the public may call on the individual or the group for the service and help that it needs.

At national level, EJC provides the connecting link between the national and the local societies or federations, which find the problems in their communities are similar to those in other communities and thus look to EJC to aid in finding an adequate solution.

EJC has a long-established and well-defined interest in policy contained in legislation affecting the engineering aspects of the development of the country, but in no instance has it taken a positive stand on a specific item of legislation. Local federations, state or regional, if formed and operated along the same pattern as EJC, would probably be permitted the same latitude of action within their fields of interest.

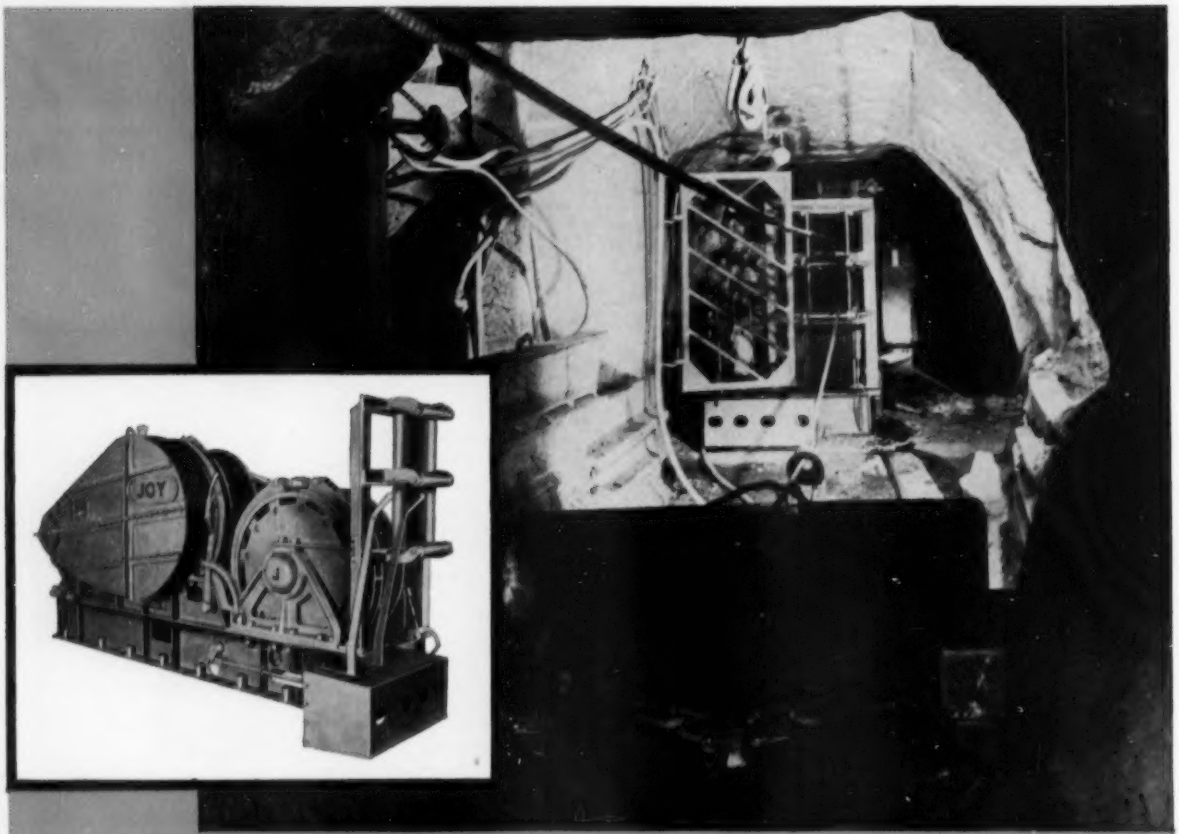
A local federation enhances its effectiveness by becoming an Affiliate Member Society of EJC. A national policy has no value unless it has the support of the local members; the individual engineer cannot know accurately what the national policy is and how it may affect the community in which he lives except through group action.

The question has often been raised concerning the advantages of dual representation in EJC. One comes through regular membership in a national society with the society board representatives speaking for their members at the EJC Board. The other comes through an observer from a federation, which has a voice at all meetings of the EJC board and a vote at the annual meeting of the board. Each serves a specific purpose. The national society board member reflects the considered opinion of the majority of the members at the national level, while the observer from the federation brings to the EJC board the specific policy problems of the area which he represents. The opinions of each are essential to the establishment of satisfactory basic policy that will be of value and benefit to the profession and the public which we serve.

Science is continuing to provide engineering with ever-expanding vistas. The utilization of new metals, the applications of nuclear phenomena, the harnessing of the vast store of solar energy, electronic and mechanical developments in automation, increasingly invest the engineering profession with social as well as technological responsibilities. The activities of this influential and representative federation of engineering groups striving to bring to bear the weight of the entire profession on these problems should stir the imagination and enlist the support of every engineer.

E. Paul Lange

(Ed. Note: Excerpted from an address presented by the author, EJC Secretary, at the Pacific Southwest Council of Local Sections of ASCE, April 19, 1956. Background information on "The Role of EJC in the Profession" appeared in MINING ENGINEERING *DRIFT, April 1956.)*



Loading and Transportation Symposium

German Bucket Wheel Excavators and Belt Conveyors

W. H. Wamsley

USED in combination with conveyor belt haulage, bucket wheel excavators offer unusual possibilities for low excavation and haulage costs. Originating in Germany, these machines are now in use or on order for Australia, Northern Rhodesia, the Belgian Congo, Italy, Yugoslavia, France, Belgium, Indonesia, and the Union of South Africa. A diamond mine in South Africa that has been using German bucket wheel excavators since 1928 purchased American cable-way excavators and American scraping equipment after the war. Operating cost for this equipment was double that for the bucket wheel machines operating in the same mines with crawler-mounted stackers.

Machines built and operated in the U. S. are, as far as the writer knows, confined to the northern Illinois coal field, where they are used in conjunction with large shovels to strip overburden too high to be reached by the 30 to 40-yd shovels. Their long stacker belts can deposit the spoil far enough behind the coal face to avoid slides, in such a manner as to make reclaiming the land easier. These large, specialized machines are now being made by Bucyrus-Erie Co. but are not adaptable to many open pit mining situations.

W. H. WAMSLEY is Superintendent at Pacific Coast Borax Co., Div. of U. S. Borax & Chemical Corp., Boron, Calif.

In Germany bucket wheel excavators are used in easy digging sand, gravel, shale, and clays overlying extensive brown coal deposits. Coal up to 200 ft thick and overburden up to 600 ft thick are being excavated almost entirely by bucket wheel excavators or by the bucket chain excavators that preceded them. Haulage that has been largely by rail is changing over to belt conveyors.

Since they were first extensively used during the 1930's, bucket wheel excavators have undergone rapid development, especially in the past ten years. They are simple and rugged and offer high output

Lignite pit with bucket wheel excavator. This unit is capable of cutting a face 16 ft below level and 40 ft above level and has a 65 ft swinging loading belt.

for their weight. Their continuous digging action causes less stress and results in lower maintenance and power costs than are achieved with shovels and draglines. Machines are available with capacities from 180 to 10,000 cu yd per hr. Digging heights range up to 230 ft and some machines are designed to dig very impressive depths below the level of their tracks. Digging widths are as much as 328 ft. One new mine in the Rhineland will operate four of the 10,000-cu yd machines. A system of power and briquette plants in the same area uses 85,000 tpd of brown coal produced by three mines.

Development and rapidly expanding use of the bucket wheel excavator have caused some design problems ingeniously solved by German engineers. A modified three-point suspension has given stability and self-equalization to the crawler undercarriage. The bucket wheel drive is a planetary gear train inside the bucket wheel itself, arranged to give three speeds—all with the same torque—selected from the control panel while the machine is in operation. Speed change is effected by an auxiliary motor driving a differential gear in the same direction as the main drive or in the opposite direction. The drive is protected from shock loads by nonfriction overload clutches which may be reset from the operator's cab. Hydraulic and electromagnetic couplings are also used. The changing center of gravity resulting from movement of the bucket wheel and discharge conveyor from side to side is automatically compensated by a simple movable counterweight. The bucket wheel without cells or internal chutes used on the American machines and trenchers was adopted especially for wheels of smaller diameter and for those required to dig below their operating levels.

These wheels without cells make possible more bucket discharges per minute and permit the buckets to be reversed for downward digging. To allow the conveyor belt to operate when the boom is inclined downward 30°, a cover belt operating at the same speed was applied. This cover belt can be lifted and disengaged when not required.

It can readily be seen that continuous discharge of materials from a bucket wheel excavator is ideally suited to the continuous haulage provided by a belt conveyor. The combination has the added advantage of discharging from the excavator material that is fine enough to go directly on the belt without crushing.

It was logical that expanding use of the continuous excavator should result in improved design. German belt conveyors permit higher speed than those of American design and are therefore narrower and cheaper; 787 fpm is common for long belts, and short belts on stackers are driven up to 1575 fpm. Troughing idlers at 30° give more capacity and better training than the 20° to 22½° standard in this country, and supporting framework is of simpler design. Conveyors are commonly constructed by fastening tubular idler supports between rails on widely spaced ties, and a bulldozer equipped with boom and roller arrangement makes shifting easy. Rails are also used to carry traveling hoppers and belt trippers.

Some mines use the same belt system for both coal and overburden. Long belts are often fed by a short intermediate belt to reduce wear on the long expensive belt and also to accelerate material to a speed approaching that of the main belt. Drives are sometimes applied at both ends. Training idlers are applied to the return as well as the loaded section of the belt. Much attention is given to belt cleaning and to centralizing the flow of materials. Limit switches are used to stop the belt in case it travels off center owing to misloading, wind, or rain. To reduce the frequency of belt shifting, crawler-mounted belts up to 200 ft long are often used.

References

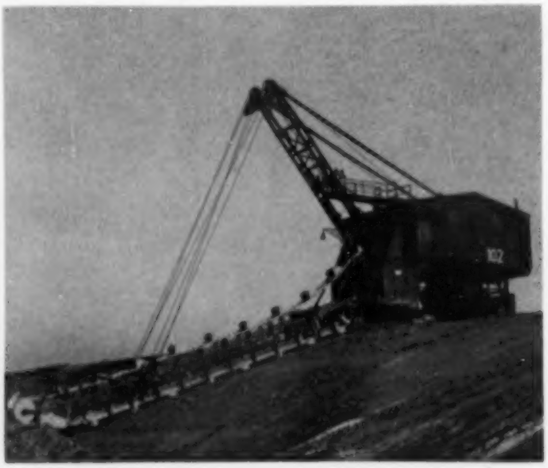
L. Rasper: Development of Bucket Wheel Excavators in Germany.

L. Rasper: The Bucket Well Excavator for High as Well as Deep Work.

W. Scharf: The Conveyor Belt System Working with the Bayerische Braunkholen-Industrie A. G.

W. Scharf: Recent Experience Gained with the Conveyor Belt System Working with the Bayerische Braunkohlen-Industrie A. G. at Schwandorf.

R. G. Weavind: The Massive Scale of Operation at the Consolidated Diamond Mines of Southwest Africa.



LEFT: High and deep cutting bucket wheel excavator with wheel at lowest position; the bucket wheel has no cells and discharges onto a rotary plate. ABOVE: Bucket chain excavators at a brown coal mine in the Rhineland. Photographs copyright by Orenstein-Koppel und Lübecker Maschinenbau A.G., Lübeck.

Loading and Transportation Symposium

Trona Mine of Intermountain Chemical Co.

Robert F. Love

DEEP under the hills of southwestern Wyoming lies a nonmetallic mineral deposit of incredible dimensions and value. This nearly pure, horizontal bed of the mineral trona, a sodium sesquicarbonate ($Na_2HCO_3 \cdot NaHCO_3 \cdot H_2O$), is found in the Green River formation some 1500 ft below surface. It is believed that evaporation of an ancient saline lake left this sodium carbonate as the evaporite and mysteriously left practically no sodium chloride, which is the common constituent of an evaporated saline lake. The trona bed averages 10 ft in thickness and extends over an area of several hundred square miles.

In 1938 a geologist with the U. S. Geological Survey identified the trona in the core from a former oil exploration drillhole. By so doing, he initiated the developments of a chemical industry of impressive growth in the area, an industry on the threshold of further and much more spectacular growth. A drilling program by the Union Pacific Rd. and Intermountain Chemical Co. in the early 1940's succeeded in blocking out mineable reserves of several hundred million tons. Development of the deposit started in 1946 with the sinking of a 12-ft diam, concrete-lined shaft 1500 ft deep—followed by construction of a small pilot calcining plant. Actual mining of the trona started in 1947 and continued on a limited scale until 1953, when the $15 million surface facilities were completed, along with the second shaft and other expanded underground facilities. Since April 1953 the mine has been producing more than half a million tons per year, from which the refining plant has produced more than 300,000 tons per year of high quality soda ash.

At Westvaco there is a modern mine with the finest known equipment, mining methods, and ventilation system. Many a mine operator saddled with an old mine, which has passed through years of operation by people with varying theories and techniques, has wished that he could start from scratch and develop the mine in a logical and proper sequence. Westvaco is fortunate in having a new mine. This mine was laid out on paper and, so far, has not varied greatly from the basic plan. During the past 10 years, however, equipment and methods have been modified and developed to keep abreast of modern trends in efficiency improvements and cost reduction. This article is concerned specifically with this phase of Westvaco's ore loading and haulage operation.

R. F. LOVE is Mine Superintendent, Intermountain Chemical Co., Green River, Wyo.

The mine now has three vertical shafts, a six or seven-entry system on the main development and a four-entry system on the panel, or secondary development. When construction work, now under way, is completed in a few weeks, the No. 3 shaft will serve as the intake airway and No. 1 and 2 shafts as the exhausts. Two 5-ft diam axial flow fans, to be located at the collar of the No. 3 shaft, will be capable of producing 300,000 cfm at 8-in. water gage pressure. A conventional room and pillar system with total pillar recovery is employed. All workings are normally carried at an 8-ft height. Rooms are 20 ft wide, 250 ft long, and on 65-ft centers. Pillars are extracted on the retreat by a system of split pillars and a protective fender between the cave and the mining face. The mining blocks, or panels, are laid out on each side of the main development at 500-ft intervals and are ½ mile long. A bleeder system of entries on the extremities of the mining areas provides unidirectional flow of ventilating air through the mining area. Methane gas is produced from the caving of the sediments overlying the deposit so that use of permissible equipment and other precautions necessary in a gaseous mine are required. Electrical power is generated at the 6000-kw power house on the surface and is transmitted into the mine at 4160 v and transformed to 220-v ac power at the face—the maximum voltage allowed underground in Wyoming. Haulage equipment operates on 250-v dc power, which is produced from motor generator sets.

Mining equipment used at the face was primarily designed for coal mining and has been modified to handle this material, which is heavier and harder than coal. Equipment and appurtenances in use include: Joy 10RU universal cutters; Goodman 665 loading machines; Joy 10SC shuttle cars; Joy RBD-11 roof bolting machines; and 5-ft long, ¾-in. diam bolts with Pattin expansion shell anchors (all roof support is by roof bolts). Main line ore haulage is by belt conveyors 36 in. wide running from the working areas to the bottom of the No. 2 shaft. A single roll crusher and automatic hoisting with two skips complete the mine haulage system.

Caterpillar, D-2, diesel bulldozer.

Model 665 Goodman loading machine dumping into a Joy shuttle car.

Ore loading at the face has progressed from hand mucking to an Eimco loader, a Joy 8BU loader, and, finally, the Goodman Model 665 loading machine in use today.

The Goodman loader is a conventional coal loading machine slightly modified to cope with the heavier and more abrasive trona. The machine is *cat*-mounted, with swinging head and tail. Speed of the gathering chains and the flight conveyor has been reduced, side clutches on the drive for each gathering chain have been added, and a special heavy duty gathering chain has been adopted. The principal advantages realized with this loader have been high capacity and improved safety for the operator. This machine is capable of loading more than 8 tons per min when working in a good muck pile. The swinging head, the comparatively low silhouette, and the location of the operator—some 12 ft back of the digging end of the machine—render this machine adaptable to the Westvaco operation.

Outage due to mechanical failure in Westvaco's loading operation has always been a disadvantage. A preventive maintenance program enables a maintenance crew to inspect and maintain a loading machine for one shift after every two shifts of operation. This regular service, in combination with scheduled major overhauls, has paid off handsomely in reduced outage. Mechanical breakdown outage of the loading operation has been reduced from an average 1.5 hr per shift in 1953 to 0.3 hr per shift in 1955.

A decision to make this a trackless mine was reached shortly after the first work in the trona bed was completed. Initial development from the shaft bottom had been with track, end-dump cars, and a Mancha trammer. This was removed in 1948 and a Joy 10SC shuttle car was installed; rubber-tired equipment has been used ever since. As the faces progressed and the length of the shuttle car haul to the shaft bottom became excessive, a 36-in. heavy duty belt conveyor was installed and was extended as necessary to keep the car haul to a minimum. There are now nine 10SC shuttle cars and more than 12,000 ft of belt conveyor in use.

Normally two shuttle cars are used with each loading machine—one right-hand drive car and one left-hand drive car—however, there have been times when the use of three cars with one loader has paid off. Use of opposite drive shuttle cars working as a team is an advantage in keeping the trailing cable of one car in the clear of the following car. The length of haul with the shuttle cars is kept to a minimum (usually not more than 500 ft) by extension and retraction of the secondary conveyor belts, which are installed in 260-ft increments as development advances and are removed in the same increments as the extraction mining retreats from the limit of the mining area.

The shuttle cars haul approximately 8½ tons per trip, and a unit of one loader and two cars can produce more than 600 tons per shift. It has been found that the secret of high production with the face equipment (loading and haulage) is to keep the operating time of the loader at a maximum. This is accomplished by a short haul, a quick dump, no delays en route (which means proper routing of cars), enough cars to work efficiently, a minimum of delays at the loader (no lump breaking, sprinkling, barring down, or ventilation work by the operator), and proper ventilation and power facilities. Time study work on the loading operation during the past few years has improved the efficiency of this operation and consequently increased production by as much as 50 pct.

The 10SC shuttle car has worked out well at Westvaco. A few modifications have improved the face haulage: 1) side boards 6 in. high have been installed on all cars, increasing payload by more than a ton; 2) a smaller trailing cable (a two-conductor, size No. 4) has made it possible to get approximately 700 ft of cable on the cable reel; and 3) a spring-loaded shock absorber on the anchor points of the trailing cable reduces cable abuse.

A major improvement to face haulage has come from the changes made in car-dumping points at the belt conveyors. The development cross-cut pattern in the room and pillar area is such that every room has a straight shot at the conveyor. In other words, there is a side-dumping point for each room.

The decision to make Westvaco's mine completely trackless was reached after careful study and analysis. Initial cost of track haulage was considerably below that of a belt haulage system; however, the operating cost of 1½¢ per ton-mile for conveyors vs 5¢ per ton-mile for conventional rail haulage, applied to the tonnages to be handled, justified the larger investment for a belt system. Mines that have a large capital investment in track haulage may find it difficult to justify the change to belt haulage, but here again with a new mine it was possible to choose the superior system.

The mainline haulage system is composed of two

Goodman type 99 heavy duty belt conveyors with 6-ply, nylon or rayon fabric, 36-in. wide belts with vulcanized splices. Idlers are 5-in. diam with sealed bearings and are on 3-ft centers. Each conveyor is some 3000 ft long, speed is 300 fpm, and carrying capacity is 450 tph. All belts have snub-drive head pulleys with gravity take-ups. Drive motors on all conveyors are 60-hp, wound rotor type; fluid couplings are used on the drive of the Goodman conveyors to improve the motor starting torque curve.

Secondary haulage from the working areas to the mainline is by Continental Gin belt conveyors with 5-ply, rayon fabric, 36-in. wide belts. Speed and carrying capacity of these secondary belts is the same as that of mainline belts. There are four of these Continental Gin conveyors, each capable of being extended ½ mile.

At the end of the mainline conveyors, at the No. 2 shaft bottom, there is a single roll, 21x60-in. McClanahan and Stone crusher, which reduces run of mine ore to −3 in. Crushing takes place just before the ore is conveyed to the 800-ton storage pocket, and this sizing is necessary to permit the proper functioning of the automatic hoisting equipment.

In the Westvaco mine, run of mine ore is transported from the face to the shaft bottom, a distance of approximately 1½ miles, over a series of belts and transfer points. Lump size has always been a big problem and efforts are made to control the size at the working face, but lumps weighing close to 200 lb are still being handled on the belts. Because the ore is relatively heavy and somewhat abrasive, there is always the possibility of abuse to the equipment resulting from improperly designed dumping points and transfer points, the lump size being transported, the alignment of the conveyor framework, and the upkeep of the entire system.

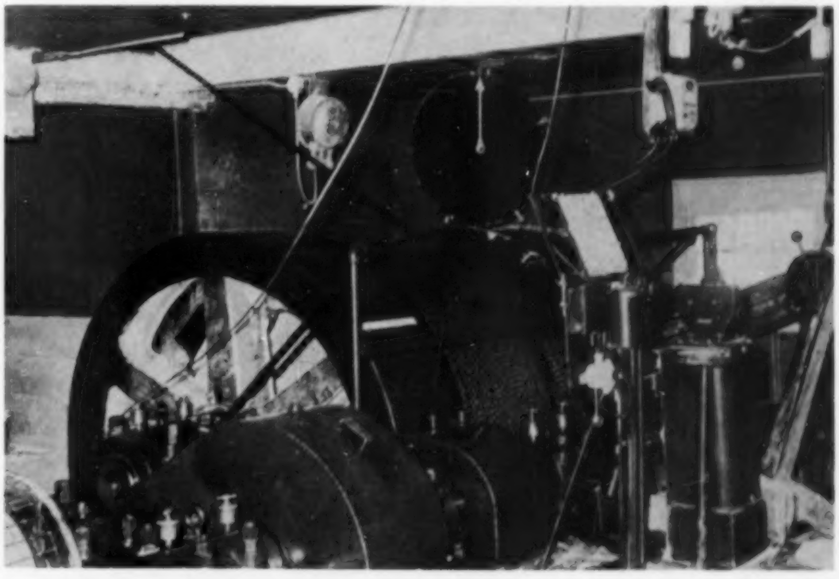
Through the years improvements have been made in the conveying system, so that the operation has become close to automatic, with the necessary protective devices to prevent damage to this expensive equipment. Manpower requirements for conveying, crushing, and skip loading equipment have been reduced from six to two men per shift as the result of improvements made. This amounts to some $60,000 per year saved in operating labor.

Improvements in the Conveying System: The point on the conveyor where the shuttle car discharges its load, called a side-dump, is composed of a hinged metal side board with a rubber skirt on the edge. This board is located on the side of the belt opposite the discharging shuttle car. It is lowered into position by a pull-cord operated by the car driver from his seat and is swung automatically out of the way when not in use. It has been found that using only one side board is better than using one on each side because the discharging car can get much closer to the belt and thus reduce the distance for the ore to drop onto the belt. Impact idlers are used under the discharge point. The side dump station is made so that it can be clamped to the belt framework at any point; this facilitates the installation of these stations.

Secondary belts have several protective devices—most of them are commonly used in conveying systems. The tail pulley is of the ribbed, self-cleaning design. A rubber-edged plow is located on the return belt, just ahead of the tail, to remove any material from the return belt. The drive motor has the usual protective devices, plus interlocked electrical controls operating from centrifugal switches to insure proper order of starting and stopping of the belts in series. These centrifugal switches also shut the belt off if slippage at the drive pulley occurs. In addition, there are shut-off switches operating from counter-balanced idlers located just behind the belt drive pulleys which permit an empty belt to keep running even though the belt ahead is shut down. A loaded belt, however, is stopped if the belt ahead is shut down. This is a tremendous advantage in multiple belt operation, since production losses at the working face, due to belt outage, are kept to a minimum.

The point where one belt discharges onto another is very critical in Westvaco's haulage system. Usually the direction of travel of the ore is changed at this point and some method of reducing abuse and preventing hang-ups must be devised. At one time, transfer point men were required at all these points—mainly to protect the belts from damage. These men have now been eliminated by the addition of a small, inexpensive device, made by Hewitt-Robins,

No. 1 shaft, which is used for men and material, is 12 ft diam and equipped with this single-drum hoist driven by a 250-hp motor.



called a Robintronic—an electronic instrument that signals when a chute is plugged and automatically shuts off a belt before any damage can occur. Transfer points have to be made as *plug proof* as possible, but with this device the worry of damage to the belt is practically eliminated. The permanent transfer points on the mainline conveyors are built with grizzly chutes to screen out the fines to the lower belt, thereby cushioning the impact of the larger material. Impact idlers of the zero-pressure pneumatic design are used under all transfer points. On the temporary transfer points, where the secondary belts discharge on the mainline, a design is being tested in which the grizzly is eliminated and the secondary belt heads are lowered to the closest point possible above the main belt. Possibility of damage from impact is increased, but possibility of a *plug-up* is eliminated.

The roll crusher at the end of the conveying system is equipped with two devices that have eliminated the need for a man on constant duty. One is the same Robintronic that signals a *plug-up* in the crusher; the other is a tramp-iron detector that stops the belt if iron is detected on the belt.

A few other improvements in the haulage system may be of interest: a two-wire 24-v *squeeze* system parallels all conveyors. When shorted together, two bare wires about 6 in. apart, suspended from the roof, stop the belt. This is an advantage to inspectors and maintenance men and provides the protection of an emergency stop at any point.

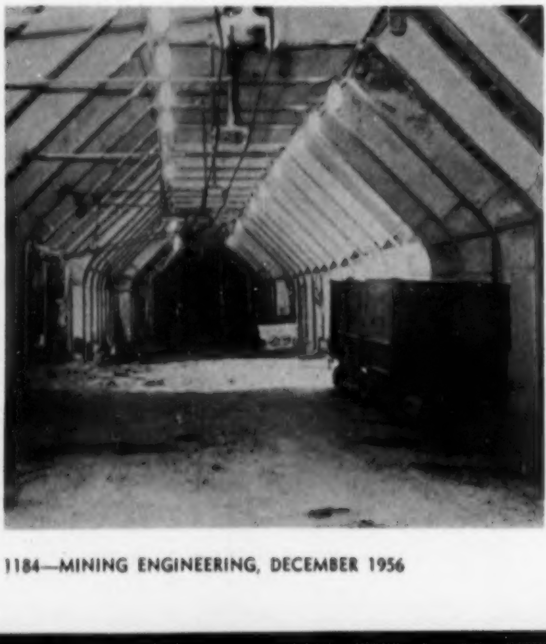
Portions of the conveyors have been hung from roof bolts. This practice has proved satisfactory but has been discontinued because of the added expense of installation. In mines where floor heaving difficulties are encountered, this type of conveyor installation is recommended.

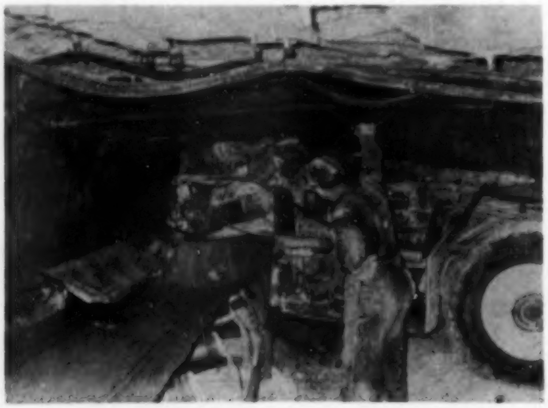
Since the secondary conveyors are constantly being extended and then retracted as mining progresses, it is important that a technique be developed to provide for maximum efficiency in this operation. Westvaco's conveyor is made in 12-ft long sections that have a quick-lock, button hook connection for quick assembly. The tail piece of the conveyor is made to ride on the side rails so that belt can be added easily by pulling the tail piece along the newly installed side rails. An electric impact wrench is used for all bolt tightening and loosening in the framework assembly. A powered belt-winder that can reel up a 600-ft length of conveyor is used. A special cart lifts the belting so that the idlers can be quickly inserted after the side rails and the belting have been extended. Normally six men can extend a secondary conveyor a distance of 260 ft in one shift.

The mine is presently served by three vertical, circular, concrete-lined shafts about 1500 ft deep. No. 1 shaft, used for men and material, is 12 ft diam and equipped with a single-drum hoist driven by a 250-hp motor and a double-deck man-cage. No. 2 shaft, used for ore hoisting, is a two-compartment, 14-ft diam opening and is equipped with a double-drum Nordberg automatic hoist with 13/8-in. ropes and Connersville bottom dump skips. No. 3 shaft, recently completed by the Dravo Corp., is an 18-ft diam opening and is presently a bare shaft with a small construction hoist and inspection cage.

No. 1 shaft, constructed in 1947, has the conventional single-drum hoist installation with 6x6-in. mahogany guides and E Long safety dogs on the cage. The unusual or different feature in this area consists of a sound-powered phone system—one phone in the cage and the other in the hoist house. These phones are regular U. S. Instrument phones, connected by conductors built into the core of the rope and a collector ring on the hoist drum. This set-up serves better than more expensive electronic equipment. The ability of the internal conductors to maintain their continuity is, of course, the secret of success. The Roebling rope in use has been operating satisfactorily for seven months at this writing. Another feature of note in the No. 1 shaft is the system of cleaning the shaft walls of the buildup of recrystallized trona from the brine seepage through the concrete lining. This buildup constitutes a serious hazard and must be eliminated. In the past years, these deposits were scraped from the walls and steel. Circular rings of perforated pipe are now installed every 200 ft in the shaft throughout the section where the buildup formerly occurred. Circulating fresh water through this sprinkling system at regular intervals of a few hours every few days has eliminated this problem.

In the No. 2 shaft completed in 1953, there is a modern, automatic skip-loading and hoisting system engineered by the Allen-Garcia Co. This system





LEFT: No. 1 shaft station showing small material trailers. ABOVE: Joy shuttle car dumping at side dump station on conveyor.

consists of a Nordberg hoist with a 500-hp dc motor; controls are of the Westinghouse Rotorol type; hoisting speed is 900 fpm with a 10-sec acceleration and deceleration period; skips run on 90-lb rail guides and are bottom dump with a 6½-ton payload. The shaft bottom works, where the skips are loaded automatically, consist of two pan-feeders drawing out of the bottom of the 800-ton ore pocket and two weighhoppers which are filled by the feeders. The weight of ore put into these weighhoppers is controlled by Streeter-Amet electronic scales. A descending skip contacts a limit switch which starts the automatic loading of the skip and the refilling of the weighhopper. Because of the water in the shaft, outage of electrical devices located in the shaft bottom was excessive. It was found that filling the limit switch boxes with transformer oil eliminated grounding and shorting in this critical equipment.

Internal conductors are also built into the ropes on this hoist. This electrical connection between the skips and the hoist house is used for voice communication for work in the shaft, plus a slack-wire indicator that shuts down the hoist if the weight is removed from the rope. This slack-wire indicator prevents damage to the rope or shaft from a *hung* skip in the shaft. Sprinklers are also used in this shaft to keep the walls clean.

Westvaco's No. 3 shaft was constructed to serve as a ventilating airway. Two Jeffrey 5-ft fans and Dravo direct-fired gas heaters are now being installed at the collar of this shaft to provide some 300,000 cfm of air at a temperature that will prevent ice formation in the shaft. The arctic climate requires this heating during several months of the winter. Eventually this shaft will be used as main man and material shaft and has been constructed with this in mind.

Personnel and Supply Transportation: Aside from the man haulage in the No. 1 shaft, personnel transportation underground for years consisted of walking. As the distance to the working areas increased, it became apparent that some mechanical means of personnel transportation was required.

The first method considered was to reverse the conveyor belts and transport the men to them. This, of course, is done in many mines. This method was vetoed for several reasons:

1) It was not considered as safe as some other forms of transportation. Where men have to load and unload several times on a trip the hazard is considered excessive.

2) It was too slow. Westvaco belts travel at 300 fpm, which means that the average trip would take approximately 30 minutes.

3) Since all the conveyors have gravity take-ups just behind the head sections, it is almost impossible to run the belts in reverse because of the belt slippage at the head pulley.

4) The belts would have to be emptied of ore at the end of every shift, which would mean a loss in production. Also, there would be no form of personnel or supply transportation at any times other than the start and end of each shift.

Track haulage for personnel and supplies was also considered, but since track was not used for ore haulage, the capital outlay was too great.

Rubber-tired haulage was finally decided upon and a mine jeep was purchased. This machine proved mechanically unsatisfactory and subsequently three Willys jeeps were converted to electrically-powered machines which ran off the trolley. These jeeps have 10-hp dc motors mounted where the original gasoline engines were. Performance of these homemade machines has been quite satisfactory. They travel at speeds up to 10 mph and seem to have sufficient power for the company needs.

Rubber-tired trailers are used with these jeeps. Two-wheel man-trailers haul a crew of 16 men comfortably. Small four-wheel trailers are used for supplies. These supply trailers are purposely of short length so that they will fit into the man-cage. Thus trailers can be loaded on the surface and taken down the shaft and into the working or storage areas. A jeep sometimes hauls a trainload of three or four trailers. Principal supplies consist of roof bolts and powder, and these are nicely handled in trailers.

Construction and maintenance of roadways constitute a big problem. If speeds up to 10 mph are to be expected, then roads must be kept in top condition. Road surfaces are oiled and maintained with a Caterpillar D-2 diesel bulldozer. Continual travel and maintenance have improved mainline roadways to the condition of a macadam road.

Use of diesel underground has been limited to the one D-2 bulldozer. A U. S. Bureau of Mines approved scrubber on the exhaust has rendered exhaust fumes only slightly noticeable and completely harmless. The flexibility and the low-cost operation have been welcomed. A diesel-powered jeep for supply haulage has been ordered and should be in operation soon. With proper maintenance of the engine and scrubber, plus regular testing of exhaust gases for CO, plus the proper quantity of ventilating air, a diesel engine can be used to good advantage in most mines.

In addition to the jeeps and trailers, the small three-wheeled, battery-powered scooters for foremen, engineers, and maintenance personnel transportation have been adopted. The model made by the Laher Corp. is best suited to Westvaco's needs. These fast, easily maneuvered machines have been of tremendous value.

Conclusions

Loading and haulage at the Westvaco mine are similar to that in most trackless mines in a flat seam, but efficiency has been improved constantly. During the past three years production has risen from 500 to 2500 tpd, and overall efficiency has improved from 8 to 21 tons per manshift. Much of this improvement has come in the loading and hauling operations.

Advances are rapidly being made in loading and haulage methods; all improvements are towards the ultimate of a continuous and automatic process. The continuous miners and extendable conveyors used in coal and potash mining are a great step forward; however, continued improvements in engineering and materials will outdate some of this equipment a few years from now.

Westvaco has a modern and efficient mine, but already much of the equipment is becoming antiquated by newer, more efficient methods. The mine research department is actively pursuing the continuous mining of trona. Continuous mining machines presently available are untried in trona, but it is believed that a satisfactory machine can be designed. A test machine will be working in trona sometime in the near future.

Loading and Transportation Symposium

Loading and Haulage at The Indian Creek Mine

Laurence W. Casteel

INDIAN CREEK MINE, newest of the St. Joseph Lead Co. properties, is located in southeast Missouri on the flanks of a buried porphyry ridge. Here the orebodies are very irregular in size and shape and have a regional pitch of about 15°, which may vary locally from 0 to 90°.

To maintain an economical grade of ore, selective mining is imperative. The room and casual pillar method is employed with 20-ft diam pillars 35 ft apart where possible. Stopes vary in height from 10 to 55 ft. All are too small to supply a loading unit continuously.

With these conditions, the loading and transporting equipment must be extremely versatile. Mobility, low operating height, and capacity to negotiate steep grades are prime requisites.

The loading unit selected as most suitable for this mine is a portable scraper ramp mounted on 8.25x15 rock lug tires. Two 10-hp electric motors with speed reducers furnish power for the four-wheel chain drive. A 2-hp hydraulic pump mounted on the chassis supplies two sets of hydraulic jacks. One set of these jacks serves to raise and lower the tail of the ramp slide, which is hinged to the chassis near its center so that as the tail of the slide goes up, the front of the slide goes down. The slide is constructed of ¾-in. steel plate with 15-in. channel iron sides. Five 3x½-in. wearing strips are welded to the bottom. When in operating position, the front 6 ft 8 in. section of the slide slopes 26° from horizontal. The next 7 ft 4 in. section is 13°, and the last section, 14 ft, is 6°. The front 20 ft of slide is 6 ft wide. The last 8 ft tapers in to 9½ in. and has no bottom, merely serving as an anchor for the pull rope sheave.

The hoist or slusher used on this ramp is a 30-hp, electric, 3-drum type with air-operated clutches. It is mounted 4 ft over the front of the slide on a steel plate hinged at the back to a supporting superstructure of heavy angle iron. At the front the plate is attached to two hydraulic jacks, which allow the hoist to be raised or lowered, depending on the height of ground in which it is operating.

The center drum of the hoist is used for the pull rope that is taken off the top of the drum, through the back of the hoist, and around a sheave anchored at the back of the slide before being attached to the scraper. Outside drums on the hoist are used for the tail ropes, which are taken off the bottoms of the drums, through the front of the hoist, and around sheaves anchored in the face of the stope before being attached to the scraper.

A 54-in. semibox type of scraper that holds about 9/10ths of one ton of ore has been found to be most suitable; ⅝-in. steel wire rope and 10-in. sheaves are used. Rollers are mounted around the front of the slide to protect the ropes. Slings and wedges provide anchorage for the tail sheaves.

The 440-v switches necessary to operate the various motors are mounted on the chassis. A small, dry transformer for the floodlights is also mounted there.

Overall length of the ramp is 28 ft 2 in., height 8 ft 9 in., and width 7 ft 6 in. The operator stands on either side near the front where he manipulates the hoist clutches by remote control.

This unit travels under its own power at the rate of 1 mph, and will negotiate grades up to 20 pct. For long moves it can be put into free wheeling and towed by a truck or tractor. Under average conditions, this ramp will load at the rate of 3 tons per min. It can reach and load rock on slopes too steep for ordinary haulage.

Haulage at Indian Creek is completely trackless. During the development of the mine when haulage distances were short, Joy 60E 12-ton shuttle cars were provided, and some are still in use on hauls under 1000 ft. With installation of a double, inverted trolley with a slide to which the cable is attached, these cars can be used for long hauls, but their slow speed of 4 mph makes this uneconomical. One shuttle car was converted to diesel-electric power by replacing the reel with a diesel motor and generator. This is a satisfactory unit, but only for short distances, since the speed was not increased.

As the mining area enlarged and haulage distances lengthened, gradual conversion to diesel truck haulage became necessary. Faster haulage was the primary reason for the change from shuttle cars, but simpler and cheaper maintenance, greater versatility, and lower initial cost were all factors.

The only truck available low enough for the Indian Creek operation was a 6-ton diesel truck designed by St. Joe engineers for use in some of the older

L. W. CASTEEL is Division Superintendent, Indian Creek Div., St. Joseph Lead Co., Potosi, Mo.

mines with narrow haulageways. This truck is only 6 ft 6 in. wide, 4 ft 6 in. high, and 18 ft 6 in. long. It has a top speed of 26 mph and will negotiate 12 pct grades in second gear and 25 pct grades in low gear. The motor is a 2-71 series General Motors diesel with a twin disk fluid clutch and Cotta transmission. Dual wheels are mounted on the rear with 12-ply tires, 7.50x15, on both front and rear. Standard truck springs are used with overloads and radius arms mounted on the rear. Two 6-in. hydraulic cylinders pivoted inside the frame furnish power for dumping the bed. Heavy duty air brakes are supplied on the rear wheels only.

This truck was put into use, and time studies were made on its performance and on that of the shuttle cars. On the basis of these studies, haulage distances and costs were determined for various combinations of hauling equipment.

To maintain a daily production of 2000 tons (mill capacity) by a two-shift operation of four loading units, each unit must produce an average of 250 tons per shift. Studies indicated that a loading unit located 1000 ft from the shaft could obtain the required production by using either one shuttle car at a haulage cost of 18¢ per ton, or one 6-ton truck at a cost of 16¢ per ton. Two shuttle cars could obtain the required tonnage up to 2000 ft at a cost of 25¢ per ton, but two 6-ton trucks could do the job up to 2500 ft at a cost of 23¢ per ton.

By projecting these figures, based on actual performance, it was possible to determine haulage distances and costs when using three or four units to haul the basic requirement of 250 tons per shift. However, the costs, initial and operating, rapidly became prohibitive as the number of units increased.

By making a few assumptions, it was also determined that a 10-ton truck similar in speed to the 6-ton truck could haul 250 tons per shift a distance of 2000 ft for 17¢ per ton. Two such trucks could haul 4000 ft for 25¢ per ton. Since 70 pct of the known ore reserves at Indian Creek lie within 4000 ft of the shaft, this would seem to be an ideal haulage unit. An investigation of larger trucks was made.

Since the haulage drifts at Indian Creek are a minimum of 14 ft wide, standard width trucks could be used. The limiting dimension is the height, since maximum allowable height is 5 ft 6 in. The largest truck using standard parts that could be designed within this dimension is now being constructed. This truck will have a capacity of 10 tons and will be 20 ft 6 in. long, 8 ft 6 in. wide, and 5 ft 6 in. high. A 4-71 series G. M. diesel motor with torquematic transmission will furnish the power. The rear axle will be attached to the frame by a patented rubber block and steel pin arrangement that provides a cushion as well as strength. The front axle will be provided with sliding pad spring mounts and radius arms and 14-ply tires, 10.00x20, will be mounted all around. If this truck proves successful, it will replace the present haulage equipment at Indian Creek.

Haulage at Indian Creek is completely trackless, using units such as this 6-ton diesel truck designed by St. Joe engineers. A 10-ton of similar type is now under construction.

The loading cycle at Indian Creek begins with preparation of the rock pile by a D-4 Caterpillar bulldozer. All fly rock is dozed up to the main rock pile, and the roads in the stope are cleaned. The ramp moves up to the rock pile and the back of the ramp is jacked up hydraulically and the hoist raised to the proper position, depending on height of ground. Tail sheaves are anchored in the face of the stope with slings and wedges driven in holes drilled by the loading and hauling crew or by the drillman on the previous shift. The tail ropes are stripped off the drums and placed around the sheaves. A hose is attached to furnish compressed air for the activating cylinders on the clutch bands, and the ramp is ready for operation. A truck or shuttle car is placed under the hole in the tail of the ramp and loading starts. The rock is dragged up the ramp slide and falls through the opening into the haulage unit. Locations of the tail sheaves and of the ramp itself are changed as needed to reach more rock.

When the stope is cleaned, the scraper is pulled onto the ramp, and the tail ropes are removed from the sheaves and wound on the drums. The hoist and the back of the ramp are lowered, and the scraper is ready to move under its own power, or by towing, to another stope.

All ore is hauled directly from the loading ramp to the shaft where it is dumped through a grizzly into the pocket. For the long hauls that will be encountered in the future, it is planned to use transfer bins and large trailer trucks. The small trucks will dump into the bins, and the large trucks will make the long hauls.

The main haulage drift serving the majority of the stopes is 11 ft high and 24 ft wide. The lesser drifts are 11 ft high and 14 ft wide. Crushed rock is used to surface the roads, which are maintained by a diesel motor grader. Several types of asphalt surfacing material have been tried, but all were unsuccessful because the road bed is damp.

All diesel-powered equipment has been supplied with scrubbers for the exhaust gases. Periodic tests run on these gases show them to be well within the limits of safety. Natural ventilation provides 65,000 cfm of air, except during hot weather when it becomes necessary to run a blower at the ventilating shaft. Small blowers are used in the stopes when necessary.

Lubrication and repair of all equipment is done by the maintenance crew. To facilitate this work a new underground shop is under construction. A bridge crane, grease pit, welding shop, supply house, and tool room will be included, because in trackless open stope mining, equipment maintenance and road maintenance are of the utmost importance.

With the equipment described here, loading costs at Indian Creek for 1956 average 21¢ per ton and haulage costs average 22¢ per ton. These costs include operating labor, power, equipment maintenance and road maintenance. With a new shop and larger trucks, perhaps the future will bring considerable improvement.

Loading and Transportation Symposium

Kennecott Copper's Ray Mine Open Pit Operations

R. I. Williams

KENNECOTT Copper's mining activities in Arizona are conducted by the Ray Mines Div., located in Ray, Ariz., in the Mineral Creek mining district about 60 miles southeast of Phoenix. The Ray orebody lies in a mineralized zone belted on the east, west, and north by well defined faults. The orebody consists principally of schist with intrusions of quartz porphyry and diabase. The dominant ore mineral in the schist and quartz porphyry is chalcocite, with some native copper, and in most areas the orebody has been heavily oxidized. In the diabase the dominant ore mineral is chalcopyrite. Ore deposition in the schist portion is very erratic, with no apparent structural explanation.

In 1873 silver prospectors opened the mining district, but it was not until 1899 that an English company made the first major attempt to exploit the copper deposits. This venture failed because of improper sampling and limited development work. In 1904 D. C. Jackling, Seeley Mudd, and Philip Weiseman became interested in the Ray Copper Co. and the Gila Copper Co., which later merged as Ray Consolidated Copper Co. In 1926 the Nevada Consolidated Copper Co. took over Ray Consolidated and in turn was taken over by Kennecott Copper Corp. in 1933. The extensive program of surface and underground development and exploration begun in 1907 continued through successive changes of management and ownership to the present time.

Underground to Open Pit

The property was developed for underground mining by block caving methods, which underwent a process of evolution until underground mining was discontinued in January 1955. It was foreseen during the later war years that because of the decline in grade of ore, increased mining costs, and the heavy ground encountered, the life of the mine as an underground operation was fast drawing to a close. Because of this an extensive diamond drilling program was started in 1945 and studies were made concerning the feasibility of using modern shovel and truck haulage equipment and methods for mining the remaining ore by open pit methods. As a result of the studies, a contract was let during the latter part of 1947 for stripping overburden from the orebody for mining by open pit methods. The first phase of the stripping program was to build a diversion channel, west of what was to be the Pearl Handle pit, to lead any flood waters from Copper Canyon away from the pit.

R. I. WILLIAMS is Truck and Dozer Maintenance Foreman at the Ray Mines Div. of Kennecott Copper Corp. at Ray, Ariz.

Original Haulage Equipment

The contractor's original equipment consisted of diesel-powered 2½-cu yd shovels, served by a fleet of diesel-powered rear dump trucks of 15-yd capacity. As the work progressed power lines were erected and electric shovels were put into service. The first full electric shovel was of 2½-cu yd capacity. Later shovels were of 5 and 6-cu yd capacity. With the increase in shovel size the contractor began to use a fleet of diesel-powered rear dump trucks of 18-cu yd capacity. As some of the hauls were relatively short and level, the contractor decided to move the overburden with bottom dump trucks, drawn by a tractor powered by a 200-hp diesel engine. The contractor's experience proved that, as long as the rock was well broken and the hauls relatively flat, the bottom dump equipment would pay. However, the tractor trailer combination did not work well on adverse haulage grades exceeding 4 pct. When diesel-powered dump trucks of 34-ton capacity (22-cu yd) became available in 1949 the contractor added six of these to the existing fleet. At peak production the contractor worked three fleets of rear dump trucks, 15-yd, 18-yd, and 24-yd, and a fleet of 24-yd bottom dump trucks. Two diesel shovels with 2½-cu yd dippers were used, together with four electric shovels, one with 2½-cu yd dipper and three with 6-cu yd dippers. Overburden was stripped seven days a week on a three-shift basis until the spring of 1949, when production was curtailed to five days a week.

In 1949 small quantities of ore were encountered above the place where ore had been anticipated. Until the completion of temporary crushing facilities the ore was stockpiled so that the stripping program could proceed in an orderly fashion. In 1950 the permanent primary crusher was placed in operation and temporary crushing facilities were removed.

Until November 1951 it was the contractor's responsibility to remove the overburden and mine the ore by open pit methods. In November the company received equipment and assumed the job of removing overburden on the north side of the pit, while the contractor remained on the south side of the Pearl Handle pit, removing overburden and mining ore. In March 1952, when more shovels and trucks became available, the company took over all mining operations.

In 1952 all work in the pit was aimed toward ultimate production by open pit methods alone. Underground mining continued until January 1955, production being supplemented from the pit to maintain a combined rate of 15,000 tons. Since then all production has been by the open pit method. With the steadily increasing demand for ore from the pit it has been necessary to add constantly to the number of shovels and trucks and to improve the efficiency of all equipment.

To utilize equipment fully and maintain the stripping ratio, when the company took over in 1952 it became necessary to place the operation on a three-shift basis, each shift being supervised by one shift foreman, responsible to a general foreman.

At that time the loading units consisted of one diesel-electric shovel equipped with a 2½-cu yd dipper, two electric shovels with 5-cu yd dippers, and one electric shovel with a 6-yd dipper.

Electric Shovel Selection

There are now seven shovels in use, of which six are fully electric. The electric shovels are all designed to operate on 2300-v ac. The power distribution circuit is so designed that either the Pearl Handle pit or the West pit now under development can operate independently. Electric power is distributed through the pit by conventional power transmission lines, which are tapped where necessary, and switch houses and oil switches have been installed. Trail cable carries the power from the switch house to the shovels. The length of the trail cable is kept to a minimum to facilitate moving shovels and placing cable towers.

Dippers are of cast manganese steel. Since inception several changes in dipper design have been made to increase shovel efficiency and reduce cost. Original shovel dippers were 5½ and 6-cu yd capacity. One of the 5½-yd dippers was replaced by one of 6-cu yd capacity. Although the larger dipper did not affect loading time appreciably, it handled blocky material produced by underground subsidence much more easily. The small increase in size practically eliminated rock hanging or nesting in the dipper lip. Also the contour of the lip was redesigned to eliminate or hold to a minimum overcast or spillage during loading. These modifications have been made gradually, since it would be uneconomical to discard the spare lips, dippers, and parts in stock. When the changeover is completed, dipper capacity on electric shovels will be either 6 or 7 cu yd.

The matching of shovel dipper size to desired tonnage is given in the following table:

Dipper Capacity, Yd	Number of Passes	Load, Tons	Desired Load	Number of Passes	Load, Tons	Desired Load
6	4	36.0	35.0	6	54.0	50.0
7	3	31.5	35.0	5	52.5	50.0

Under optimum conditions haulage capacity could be fully utilized using four passes with the 6-yd dipper, but too often, with rocky material, the load required five or even six passes. Only four of the machines have sufficient power to handle the extra weight of a 7-yd dipper plus load. The other two electric shovels have only enough power to handle a 6-yd dipper and load.

The dipper teeth on the first four shovels delivered were a bolt-on or Clark type. These did not prove satisfactory for several reasons: 1) bolts and nuts were difficult to install and replace and 2) the tooth bases with the bolt hole were easily broken. Once the tooth base was broken the tooth could not be kept on the dipper and the dipper lip had to be changed. To avoid this delay and reduce repair costs a change of the lip was undertaken. The replacement lips were designed to take the Whisler or clamp-on type of tooth. In the meantime the original lips were reworked to accommodate the Whisler-type tooth. Several different Whisler-type teeth have been used. The first teeth used were of one-piece cast manganese steel construction. Several types consisting of an adapter and a wear point have also been tried, unsuccessfully, since the adapter wore as rapidly as the point. Hard-facing and build-up welding did not prove economical, so the one-piece Whisler tooth has been used almost exclusively.

Recently a new design of the adapter and tooth tip has become available. This new adapter has a replaceable wear plate that can be changed when the points are changed. To date the experience with this new tooth indicates a lower cost per tooth and less down time to change teeth.

Matching two different sizes of shovels with two different sizes of haulage units has been difficult. Solution of this problem has been met by compromise, through having 50-ton haulage units service the 7-yd shovels whenever possible. To utilize all the available shovel time more fully, trucks are loaded on both sides of the shovel. This avoids shovel delay while waiting for a truck to spot and provides time to clean up the shovel pit.

Clean Pit Proves Vital

Clean, smooth shovel pits are of prime importance. A clean pit lowers tire and maintenance cost drastically, and reduces down time caused by broken frames, springs, and damaged dump beds. To load on both sides of the shovel it has been necessary to provide either an underpass or crossover so that the heavy trucks will not damage the shovel trail cable. The first method tried was a bridge made of welded pipe. This method did not prove successful for several reasons: 1) A truck was needed to move it by lifting it with a dump bed. This necessitated close spotting by the driver and was unproductive time for the trucks. 2) The bridge required frequent spotting to enable trucks to reach the loading area. 3) The bridge was easily damaged.

The second method used was a pipe crossover. The crossover was not easily seen and required frequent moving involving much hand labor. This was an unfavorable factor from a safety standpoint and the method was discontinued.

The third method proved more successful and is now in use. Two steel towers were constructed on a platform with steel runners made from 8 or 10-in. pipe. The top of the tower holds a sheave mounted on a swivel. The power cable is clamped to the

platform and strung from sheave to sheave. In this manner it is possible to carry the cable overhead for spans up to 80 ft. The long spans make it easier for the trucks to spot and require less moving. Towers are usually moved by the bulldozers assigned to clean the shovel pit. It is also possible to move the towers without delaying the haulage trucks.

Loading Affects Bank Height

The banks in the original development program by the contractor varied from 35 to 50 ft high. The company standardized the banks at 50 ft. However, in recent months some of the banks have been reduced to 40 ft in height where ground breaks in a blocky manner. Two factors largely determine the set-in of the shovel at the loading face: degree of shovel swing and position of the loaded truck with respect to the direction of loaded haul. While it is highly desirable to hold shovel swing to a minimum to increase loading efficiency, the trucks should be loaded while headed in the direction of travel. In this way sharp turning while they are loaded is held to a minimum. There is less spillage over the sideboards and this reduces the amount of cleanup to be done by the bulldozers. The time consumed by a dozer cleaning the shovel pit is frequently needed for an empty truck to spot under the shovel. Excess spillage can also add to tire cost materially through cuts and impact breaks. Turning loaded trucks in confined areas also adds to chassis maintenance. Since at times it is impossible to satisfy both requirements, the foreman must decide the set-in to be used in order to make the most efficient use of the equipment.

Loading in confined areas is more often encountered in starting new benches and drop cuts. The planning of new benches and drop cuts takes these factors into account and every effort is made to develop the benches in such a manner that both requirements may best be satisfied.

At times it is necessary either to load larger boulders or leave them for the drilling and blasting crew. The choice is left to the shift foreman. If the decision is to load the large rock, the first two passes must be of fine material to provide a cushion for the large rock. This is done to reduce the possibility of damage to the truck box or chassis. Pieces of ore rock that cannot be passed through the shovel dipper are dumped on a stockpile. The drilling and blasting crew can then drill and blast them. In this way crusher plug-ups and delay at the loading area are avoided.

Selective Loading

As much of the area had been previously mined by underground methods, several peculiar problems have arisen. There are many pillars of high grade ore, so a close watch must be maintained on shovel faces during loading in old cave areas. This is done by a sampler, although many of the shovel operators have become proficient at judging whether the material is ore or waste. It is highly probable, because of the erratic occurrence of pillars in old caved areas, that a shovel might load four different types of material to be delivered to four different destinations. There could be ore, waste, oxide ore (to be stockpiled until completion of the leaching plant), and wood. Since each of these materials has a different destination, the shift foreman's close attention is necessary to avoid delays at the crusher or the shovel. A system of whistle signals has been devised to tell the truck drivers the contents of his load, waste, ore oxide, or wood.

Because of its capacity, the diesel shovel has been used principally as a standby unit. The mobility of the machine has made it invaluable in cases of breakdown of an electric machine. This shovel can also be used in remote parts of the operation where it would be uneconomical to construct power lines.

Late in 1954 a rubber-tired tractor-shovel was purchased. This has proved to be a most versatile and useful piece of equipment. Equipped with a 2¼-cu yd bucket, the tractor-shovel is used in a variety of jobs ranging from pipe laying to loading railroad gondolas with flux. On occasion it has even been used to clean shovel pits.

The original fleet of haulage trucks consisted of 15 diesel-powered rear dump units of 34-ton capacity. Since that time six additional units of 34-ton capacity and four units of 50-ton capacity have been added to the fleet to give a total of 25 haulage trucks. Most of the trucks of 34-ton capacity are powered by twin diesel engines, 200 hp each. The four 50-ton trucks are powered by two diesel engines, 300 hp each. The trucks are equipped with torque converters and semiautomatic transmissions. The decision to purchase trucks so equipped was made for two reasons: 1) ease of driver training and 2) anticipation of lower operating cost through reduction of axle breakage and transmission failure. Experience has shown that this decision was sound.

Turbocharger Solves Problem

In 1952 the adverse haulage grades ranged from 2 to 4 pct and all the ore was hauled on adverse grades. At that time length of haul varied from 2000 ft to 1 mile. As the pit perimeter was moved back the hauls became increasingly longer. As the length of these hauls increased, and the percentage of adverse grades also increased, it became necessary to add more haulage trucks to the fleet. An investigation to increase the efficiency of the existing fleet opened three possible avenues of action: 1) increasing truck loads at a possible sacrifice of speed on the adverse grades, 2) lightening loads to increase truck speeds, or 3) maintaining loads at a constant while increasing the speed of the truck on adverse grades. To increase efficiency of the existing fleet it was decided to increase the engine horsepower. This could be done by supercharging the engines with 1) a Roots-type blower or supercharger or 2) turbochargers. The turbocharger was chosen to conserve the engine horsepower and fuel required to drive a conventional positive displacement type of blower. Thus the entire increase in engine horsepower could be utilized to move the load.

The following data show a gain of speed and decrease in cycle time:

Speed, Mph	Length of Haul, Miles	Time Loaded, Min	Time of Return, Min	Time Cycle	Percent Saving in Cycle Time
6	2	20.0	4	24.0	
8	2	14.1	4	18.1	24.5

The increase in engine horsepower did not appreciably affect the fuel consumption as might be expected. The average increase in fuel consumption amounted to about 1½ gal per hr. The engine and power train components might be expected to operate for a shorter number of hours at the increased horse-

Summary of underground Mining operations that were discontinued in 1955.

Originally ore was mined from alternate shrinkage stopes and undercut pillars and loaded directly to hand tram cars on the sublevel or into 5-ton ore cars on the haulage levels. This method was superseded by undercutting a block above the sublevel and drawing the ore into grizzly raises spaced on 25-ft centers throughout the block. To cut down on haulage drift and raise development, single dump raises were later placed in front of each sublevel drift, or one or two raises were driven to serve a whole block. Ore from the uniformly spaced drawpoints was conveyed horizontally to the dump raises first by shaking conveyors and later by belt conveyors and slushers. The ore was hauled in trains made up of 5-ton cars and drawn by cross compound high pressure air locomotives to the hoisting shaft, where three cars at a time were dumped in a rotary dumper. It was then hoisted to a receiving pocket at the crusher, broken to $-3/4$ in., and conveyed by belt to 12,000-ton storage bins.

---

power. However, the increase in horsepower and speed has not, as of this date, adversely affected maintenance cost. It is felt that the increased engine power makes it unnecessary to operate at sustained maximum output as was the case with the naturally aspirated engines. In addition engine lugging and converter heating have been eliminated. To haul safely down favorable grades, as will be necessary when ore is mined from the West pit, the fleet had to be equipped with oil retarders, which operate separately from the air brake system. The fleet is also being equipped with safety valves to seal off broken air lines and ruptured brake diaphragms.

To keep downtime to a minimum the turbochargers and oil retarders were installed at the same time. Installation of these units was undertaken in conjunction with a rebuilding program, which included modification of many parts of the unit either to simplify maintenance or increase driver comfort and safety. The result was a truck tailored to fit the needs of the operation.

Preventive Maintenance

Ever increasing demand necessitates continuing scrutiny of each operation affecting availability of the trucks. This includes tire repair, preventive maintenance, breakdown maintenance, and time consumed in servicing trucks with fuel and lubricants. Such time is held to a minimum. The preventive maintenance program insures that each truck is completely inspected and tuned every 500 hr.

To minimize truck repair, every effort is exerted to keep roads, waste dumps, and shovel loading areas clean and smooth. Experience has shown that this extra work decreases materially the overall cost of mechanical maintenance, while increasing the overall efficiency and speed of the trucks. To date the two stabilizers tried have been lignin sulfonate and calcium chloride. Using calcium chloride has reduced drastically the amount of water necessary to lay dust on the roads. Use of lignin sulfonate presents another problem. No water is required, but since the material adversely affects the concentrating process, some method must be devised to eliminate the material before it is shipped to the concentrator. It has therefore only been used in waste areas.

In the interest of safety, divided roadways are used wherever space permits to separate loaded trucks from empty ones. During operations after dark, every effort is made to light the waste dumps and shovel loading areas as much as possible. Lights on the dumps are provided by generators driven by gasoline engines.

Eighteen trucks service three shovels per shift, the remainder either being repaired or serviced. It is necessary to give close attention to the length of hauls on each shovel so that the trucks and shovels may be used most efficiently and yet provide the required tonnage of ore at the crusher. This means close attention on the part of the shift foreman, since in actuality there are three fleets of trucks in operation at the same time: naturally aspirated 200-hp units, 250-hp turbocharged units, and 300-hp turbocharged units. Since the 34 and 50-ton turbocharged units cycle well together, they are used on hauls that will fully utilize their increased speed against adverse grades.

Ore production is concentrated on the day and afternoon shifts, the night shift providing only enough ore to make the necessary tonnage. Long waste hauls are usually assigned to the night shift. This is more efficient for two reasons: 1) the heat problem encountered in long hauls against adverse haulage grades is minimized by the lower temperatures encountered at night, and 2) the trucks required for ore production can be released for stripping for most of the night shift.

Auxiliary Equipment Raises Overall Operating Efficiency

Many equipment items are used in an effort to make the job more efficient. These include new rotary drills, which have substantially reduced drilling cost; a fleet of seven crawler tractors, two equipped with torque converters; and three rubber-tired bulldozers, one of 200 hp and two of 165 hp. Crawlers are used principally for heavy pushing, dump cleanup, road maintenance, and pioneer work. The rubber-tired dozers are used principally for shovel pit cleanup, towing cable skids, and general sweeping or cleanup.

Experience to date with tractors equipped with torque converters indicates reduced cost for clutches and transmissions.

To service the equipment and transport men and supplies, a fleet of auxiliary trucks is necessary. These trucks range in size from ½-ton pickup trucks to 3-ton trucks used as water trucks. At the present time one water truck is equipped with a pump to spray the water onto the roadway. The experiment to date shows that a better spray pattern using less water can be obtained by using the pump.

The pit mechanic, pit electrician, and sampler are provided trucks with radios. Foremen use pickup trucks with two-way radios. Radio communication has saved countless hours and greatly added to the safety of the operation.

Many improved methods and techniques now in use at the Ray Mines Div. were suggested by men on the job. Management has seen the importance of cultivating these ideas for improvement. To insure that credit and recognition are given where due, a system of communications is maintained. This offers an opportunity for every worker to express his ideas and suggestions. With this and other management tools, the Ray Mines Div. is collectively striving to insure a long and profitable life for the mine.

Loading and Transportation Symposium

Loading and Haulage for The Gismo System

Gismo system is built around two tractor-drawn interchangable machines: a loading-transporting unit, and drilling unit.

Dale I. Hayes

A COMBINED loader and transport, the Gismo is now built for sale in one size, 5 to 6 tons. It will operate in a minimum opening of 7x7 ft and can be built to operate at lower heights. Where space limitations are not a factor it could be designed with much greater carrying capacity—12, 18, or 25 tons—to be powered with a standard heavier tractor.

The Gismo is mounted on crawler tracks of the same design as the tracks of the small Allis-Chalmers HD-6B tractor used to power it. With the HD-6B tractor it will negotiate upgrades of 12 pct and downgrades of 20 pct. The next tractor size would make the device feasible for steeper grades or for a machine of larger capacity at moderate grades.

Under normal conditions the machine loads itself in less than 1 min after arriving at the foot of a muck pile. Muck piles have various shapes, influenced by character of rock, blasting procedure, and blasthole burden patterns such as *cutting in* rounds, back or rib slabbing or benching, and drawhole flows. Newly blasted muck piles, often thrown back 60 to 100 ft from the face, consist of widely scattered fragments at the toe that gradually thicken to some point near the face. The Gismo starts loading and cleaning up at the extreme toe of the muck pile and through the gradually thickening pile to the face cleanup. During this process the operator has complete control of the dipper edge and view of the bottom he is creating from the material he is loading for transport. This machine, with its single operator, can be controlled to build its own roadway and stope floor or working bottom to any desired longitudinal grade and cross-sectional shape, including ditches that may be desired. With other equipment to do the same job, several machines and operators are employed: bulldozers create a bank to shovel against, a loading machine loads the transport, the bulldozer smooths the bottom, a road grader builds and maintains the roads, and usually hand shoveling cleans up the bottom corner of the face for rock drilling. Some mines, using the many machine methods, find it necessary to complicate matters further by taking back into the mine crushed and sized road material made from material previously taken out.

D. I. HAYES is Assistant to the President, American Zinc, Lead & Smelting Co., Knoxville, Tenn.

This machine is capable of digging out of a muck pile large boulders or unblasted fragments of rock that would delay all other handling and crushing operations. The Gismo operator can remove oversize fragments from the muck pile, transport and unload them at a nearby reblasting area, and reload the fragments, from time to time, in the course of other work. Boulders exceeding 4 tons can be handled in this manner.

The machine is not crash loaded. The dipper edge is crowded into the muck pile slowly, without violent elevating and throwing, or excessive dropping of the muck. Dust dispersion and wear and tear on equipment are reduced to a minimum. This method of loading also gives the operator the needed control of his stope bottom, roadway, and grades—vital to preventive maintenance.

The Gismo loading mechanism is a mounted integral with body and an operator-controlled dumping device and is therefore a self-loading transport. The present model has a maximum transport speed of the tractor—about 390 fpm. Maximum speed is seldom needed or used.

When the one-way transport is about 250 ft, the machine loads, transports, and dumps at the rate of 100 tph. Hourly capacity increases or decreases as the transport distance is shortened or lengthened. At the Grandview mine average performance per shift was 285.99 tons in 1953, 388.92 tons in 1954, and 319.47 tons in 1955. At all times during this three-year period, shift output was limited to the full capacity of the concentrator. Haulage distance was 150 to 250 ft in 1953, 200 to 400 ft in 1954, and 300 to 900 ft in 1955. It was not uncommon, when conditions required it, for one machine to handle more than 700 tons in one shift.

A figure of 300 tons per shift (6 hr working time) is often used to determine equipment requirements under normal conditions. This provides considerable reserve capacity for getting out the shift production when unforeseen circumstances greatly limit the time the Gismo can work.

Capacity is also demonstrated by the proven ability of the machine to muck out about 100 tons from a round broken in an 11x11-ft drift in less than 45 min, with a haulage distance of 250 ft to dump muck into the skip loading pocket. It has mucked out a 150 to 165-ton round and loaded muck into a string of ramp cars in about 1½ hr when average trackless haul was about 185 ft. This machine would operate in an 8x8-ft drift at the same tonnage capacity where an 8-ft round would amount to 50 tons.

The Gismo was designed as a means of gathering up the blasted ore in a stope ore drift or tunnel heading and delivering it to a central transportation system. Without doubt, trackless equipment is best adapted to this job for limited distances, depending on many local conditions. For large tonnages taken from large orebodies, the economic limit might be 300 to 600 ft, whereas for small daily tonnages it might be most economical of labor and equipment to haul trackless as much as 1200 ft.

The Gismo can transfer its load into a mine car or low body auto truck, over a side or end ramp, or use a trackless roadway at higher elevation than rails. It would be practical, under certain conditions, to use inclined roadway slots to lower the top of a truck body, possibly to floor level. With accessory equipment, the Gismo could excavate these slots and could then stockpile the material for future handling and disposal.

A mine laid out for use of slusher scrapers and rail haulage is ideal for the Gismo system. Development of rail haulage and chute raises would be reduced one half for narrow orebodies and one eighth, or less, for wide tonnage orebodies. This method of development, where it exists or would be practical to install, permits organization of a Gismo ore rock excavating system that provides surge pocket storage, offering an overall production system that is extremely simple and most economical of labor and equipment. This is done by separating from each other the normally interlocked and closely time-cycled departments of a mining operation, such as hoisting, haulage, loading, drilling and blasting, maintenance, and general services. Such a system is built around the large capacity potential of the Gismo self-loading transport and the Gismo stope drilling jumbo. Production per man, or per machine, is greatly increased, as well as the tonnage produced from a given area, stope, or development. There is conspicuously less activity, which accounts for the great cost reduction.

Summary

Because it heads the mechanized production line, the drilling and blasting division of the mine ore extraction process is most important—and can be very costly. More often than not, it is not well supervised. Early development of the Gismo stimulated great improvement in this department. For example, two years with the Gismo four-drill stope or drift drilling jumbo, under extremely hard rock conditions, brought about the following average results for 1954 and 1955: 197.22 tons per rock drill operator shift as compared to 40 tons for the method it replaced; as compared to stope jumbo drilling, Joplin district, 80 tons; Canadian mechanized stope drill jumbo, 100 tons; southeast Missouri jumbo, 65 to 85 tons; and jacklegs, 61 tons.

Transportation—including rail haulage, loading, and dumping—was improved greatly because an improvement concept had been created. Originally 12 men operating four trains per shift, adit and surface and underground hauls, were getting out 20,000 tons a month in a six-day, double-shift operation. Soon two men operating two trains, one adit and surface and one underground, will be getting out 18,000 tons per month in a five-day, single-shift operation. Production will average 428 tons per manshift as compared to the former 64 tons per manshift.

Loading and Transportation Symposium

Kelley Mine Of Anaconda Co.

A. R. Sims

ORE from the Greater Butte Project is hoisted through the new Kelley shaft. In 1946 when plans were formulated for the Kelley mine, two test cave blocks were mined by using the facilities of the old St. Lawrence mine. These blocks, known as the Adams No. 1 and Adams No. 2, produced more than half a million tons of ore. The experiment proved beyond all doubt that cave mining was possible—and economical—in Butte.

Adams No. 1 was patterned after Inspiration cave blocks. Adams No. 2 was a slusher type, with the slusher lane on the first floor above the haulage crosscut and at right angles to it. Thirty-horsepower electric hoists slushed ore from 48-in. scraper buckets directly into cars for haulage to the shaft. These cars were the 57-cu ft Granby type hauled by 6-ton battery locomotives over 40-lb, 18-in. gage track. Prior to this time the largest slushing machines used at Butte were 15-hp electric hoists pulling 36-in. scrapers.

Inspiration gravity-type cave blocks were still thought best for Butte conditions, although the Adams No. 2 block indicated that scraper blocks were satisfactory from both a cost and a production capacity standpoint.

Early in 1955 a scraper block was tried out on the 600 level of the Kelley mine. This was a pillar block surrounded by gob on four sides. Fifty-horsepower slusher hoists and 54-in. and 60-in. slusher buckets were used. This operation was so successful that it was decided to use more of this type. The 600 level was laid out for Inspiration-type blocks. Several of these were changed to slusher blocks. Results were so encouraging that it was decided to change the entire layout of the 1300 level to blocks of the slusher type.

To date more than 2 million tons of ore have been mined by this method from 16 slusher blocks, which vary in size from 60x80 to 120x160 ft. Five of these blocks were mined without the use of a grizzly level. When the rock caves freely and very little secondary blasting is required, eliminating the grizzly level saves considerably in cost of block development.

A. R. SIMS is General Superintendent of Mines, Anaconda Co., Butte, Mont.

The slusher lane for these blocks without grizzly levels is on the second floor instead of the first floor above the haulage level. This provides room so that large boulders can be pulled out of the way and blasted later at a convenient time.

Slusher hoists of 30, 50, 60, and 75 hp have been tried. The 75-hp machines with a 60-in. bucket scraper are best suited to Butte requirements. Most of these are 440 v; some are 2300 v. The 2300-v installations cost less, as smaller conductor cable can be used.

Since the distance center to center of slusher lanes is a multiple of the overall length of the cars, two or more cars of a train are loaded simultaneously. One slusher block on the 1300 level has produced 2500 tons of ore each 24-hr period for several months. This block is 120x120 ft and has two slusher lanes. The 64 finger raises serve an area of 225 sq ft.

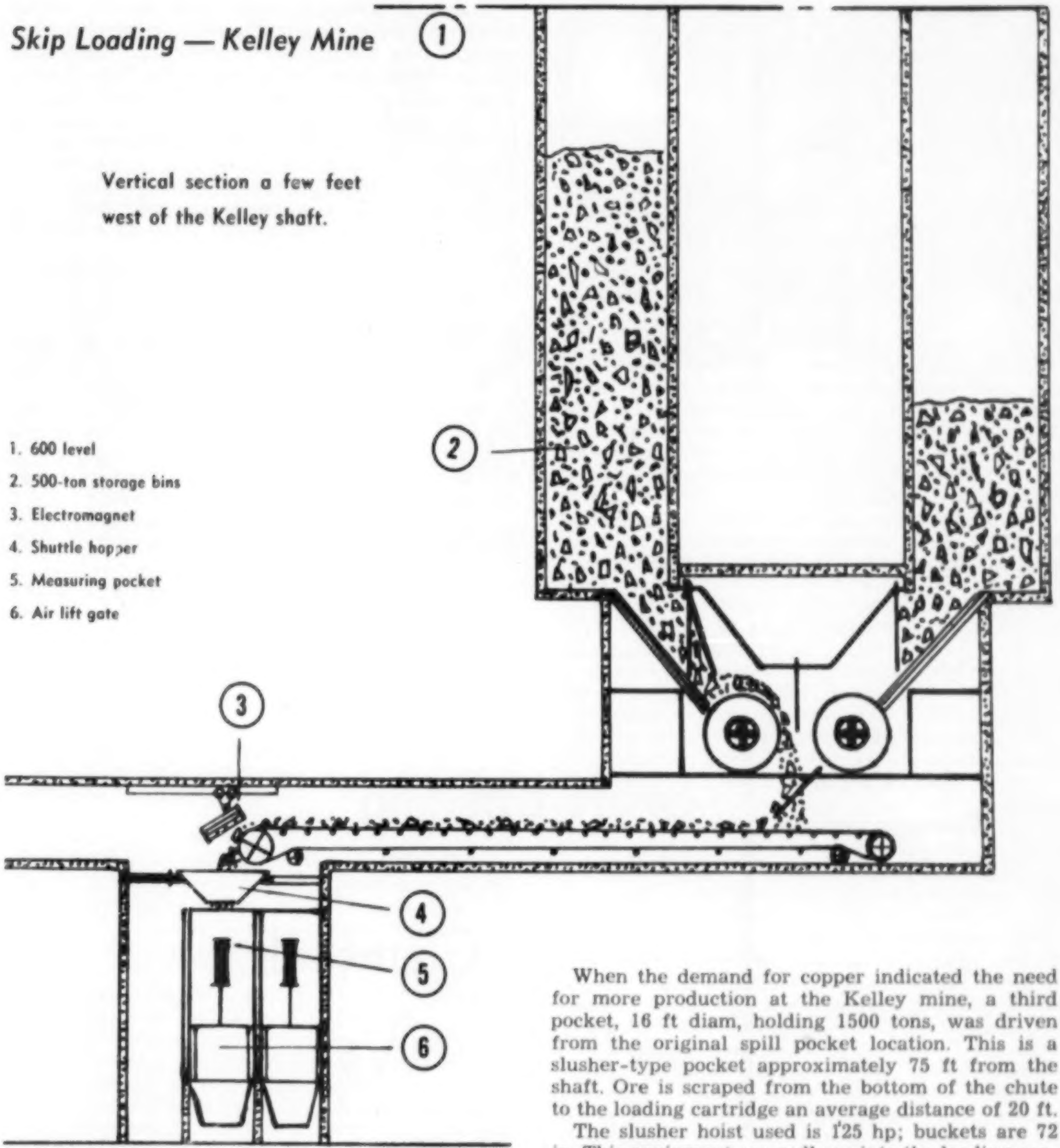
Heavy repairs on grizzly levels necessitated using concrete support in these openings, virtually eliminating repair costs. The advantages gained by having all finger raises available for production at all times, and the greatly improved ventilation in all working areas, more than offsets the much greater initial cost of the grizzly levels. Closer spacing of finger raises is also possible when grizzly levels are concreted, which contributes to a greater production with less dilution of the ore product. With improved ventilation on grizzly levels increased chute tapping efficiency was reflected immediately.

Pillar blocks have been mined by driving the slusher lane from the haulage crosscut of an adjoining mined-out block, thus saving the cost of a new crosscut. These slusher lanes are driven from chute openings used in the mined out block, so that a further saving is made in the development cost of the slusher block.

To date more than three miles of grizzly drifts and more than a mile of slusher lanes have been concreted. None of these openings has failed during the life of the block.

There is enough clay and moisture in the Kelley ore to make it difficult to handle in inclined chutes, where it builds up at the bottom until there is no room for material to flow through the chute. Slusher-type blocks greatly facilitate handling this sticky material.

Skip Loading — Kelley Mine



Vertical section a few feet west of the Kelley shaft.

1. 600 level
2. 500-ton storage bins
3. Electromagnet
4. Shuttle hopper
5. Measuring pocket
6. Air lift gate

Skip Pockets: Two skip pockets on the 600 level are located 40 and 80 ft from the center of the north hoisting compartments of the Kelley shaft. This arrangement insures sufficient ground pillar between pockets and the first pocket and the shaft. The pockets are circular sections, 14 ft in diam, lined with concrete and rails. Each has a capacity of 500 tons.

These pockets discharge onto drum feeders 6 ft in diam, which in turn discharge onto a 60-in. conveyor belt feeding into two 15-ton loading cartridges located directly in front of each hoisting shaft. An air-operated diversion hopper fills each cartridge in turn. The belt and drums are equipped with variable speed motors so that the 240-cu ft cartridge is filled while a skip load is being hoisted to surface. An electromagnet picks tramp iron off the belt.

When the demand for copper indicated the need for more production at the Kelley mine, a third pocket, 16 ft diam, holding 1500 tons, was driven from the original spill pocket location. This is a slusher-type pocket approximately 75 ft from the shaft. Ore is scraped from the bottom of the chute to the loading cartridge an average distance of 20 ft.

The slusher hoist used is 125 hp; buckets are 72 in. This equipment can pull ore into the loading cartridge faster than it can be hoisted to surface.

Cost of this installation was about one third that of the original set-up operating with drum feeders and conveyor belt. The new arrangement has proved entirely satisfactory.

Chutes: The first chutes at the Kelley mine had been used in conventional Butte mining operations for several years. Because of the sticky character of the ore they were not satisfactory at the Kelley. Arc gates of the type used at Inspiration were tried, but these too were not entirely satisfactory. Air-operated arc gates on the two main ore transfers also gave considerable trouble.

Air-operated guillotine gates have given the best results so far. These are, of course, much more expensive than any type tried, but they allow positive control of ore flow and permit greater loading speed. They may be salvaged and used in other locations many times before they are worn out.

Transportation of Ore: Ore cars are Granby type, of 117 cu ft—approximately 5-ton capacity. These are hauled in trains of 20 to 26 cars by 15-ton trolley locomotives. Tracks are 36-in. gage and of 70-lb steel.

An underground car repair shop is maintained on the 600 level, and equipment is kept in first class condition at all times.

Owing to the sticky nature of the ore, a mechanical car cleaner is installed at the top of skip pockets. It is necessary to clean the cars every third or fourth trip.

Ten trains of 20 to 26 cars each are operated on the 600 level of the Kelley. The train interval at the skip pocket is about 3 min and frequently several trains follow each other through the dump.

The haulageway is laid out in a loop so that trains run in one direction only. Each locomotive is equipped with an electronic telephone, and a train dispatcher whose office is near the shaft controls the movement of all trains. Underground men are transported from the Kelley shaft to their working areas in specially designed mine cars.

The Kelley mine produces 10,000 to 12,000 tons per 24-hr day on the 600 level. Production from this level has sometimes been as high at 15,000 tpd.

Surface Ore Bins, Crushers, and Belts: The four 240-cu ft bottom dump skips discharge into a 30-ft circular steel bin of 1200-ton capacity. These skips are equipped with 16 rubber-tired ball bearing guide wheels mounted on rubber pad shock absorbers. Two pairs of wheels at the top and two pairs at the bottom of the skip run on the face of the guides. The other wheels are mounted so as to ride on the edge of the guide. Guide wear is thus reduced to a minimum, and it is interesting to note that not a single guide was changed until after more than 13 million tons of ore had been hoisted.

A 4x15-ft apron feeder with variable speed drive directs the flow of ore from this bin onto a 6-in. grizzly. Fines drop through the grizzly directly onto the 42-in. by 320-ft conveyor belt. Coarse rock goes directly into a 36x25-in. jaw crusher, set at 6 in., which discharges onto the same conveyor belt.

This belt carries the ore up a 16° incline to the top of the railroad ore bins, where it is discharged onto another belt 36 in. by 250 ft. This second belt is equipped with an automatic self-propelled, self-reversing tripper that discharges at all points along the top of the ore bin. Ore bins, which are located 240 ft south of the shaft, have a capacity of 7500 tons.

Loading and Transportation Symposium

Shaft Loading Clamshell Vs Crawler-Mounted Loader

T. M. Berry

IN the past few years changes have taken place in shaft sinking. Progress has been made with types of machinery designed, developed, and made available to shaft sinking contractors. This is particularly true of drilling and mucking equipment. Compared with conventional clamshell loading, mucking or loading with a crawler-mounted overshot loader is a new technique for shaft work.

There are several methods of clamshell loading. Dravo Construction Co. mounts an air tugger on the clamshell to operate the closing line. The load, or holding line, is attached to the hoist of a crane or whirler at the shaft collar. The clamshell is raised or lowered by the hoist when a signal is given by a bell or light system operated by men at the bottom of the shaft. Two men control the drift of the clamshell with tag lines, pulling it to any position in the shaft for loading, or over the shaft muck bucket for emptying. One man operates the air tugger to open and close the clamshell by means of two lines attached to the operating air valve, just as reins are used to drive a horse. One man operates the signal telling the hoist operator when to raise or lower the clamshell. This method requires two hoisting lines in the shaft—the line from the mine hoist that lifts

T. M. BERRY is Superintendent, Contracting Div., Dravo Corp., Pittsburgh.

the shaft muck bucket after it is filled by the clamshell and the load line from the crane to the clamshell.

In a similar method of operation both the closing line and the load line of the clamshell are attached to hoists at the shaft collar. These hoists are remotely controlled by the operator at the bottom of the shaft and solenoid valves control the two hoists. The operation of the clamshell in filling the muck buckets is practically the same as the first method. The major difference between the two is that the second method requires three hoisting lines in the shaft. The problem of keeping these lines separated for safe operation is more complicated.

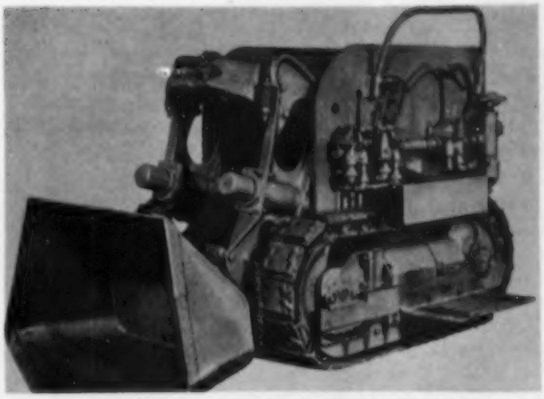
In other methods of clamshell loading, all loading mechanisms are mounted in the shaft. One method operates on the principle of a bridge crane. Supports for the bridge are located in the shaft lining or, if the shaft is unlined, they are anchored in the rock. The bridge is designed so that it can rotate in a horizontal plane for a round shaft, or travel backward or forward in a square or rectangular shaft. A two-drum hoist is mounted on the bridge. Usually the operator can maneuver the clamshell to almost any position in the shaft for mucking. This type of machine takes up considerable space in a shaft, and regardless of the size of a shaft, operating space is crowded. Lowering the bridge and supports—which must be done at periodic intervals to keep the crane close to the mucking operation—is also a task. Hoisting speed is reduced, since the muck bucket must elevate through the bridge at a slow rate of speed on its way to the collar.

Another method, similar to the bridge crane, uses a two-drum hoist to operate the clamshell, which loads into a skip. The skip is raised a short distance up the shaft to a dumping station, where a shaft muck bucket is stationed. The skip dumps into the muck bucket, which is then taken to the collar.

A newly developed method mounts a hydraulically controlled clamshell on a hydraulically controlled boom beneath the operator's cage or platform. Control is so precise it is similar to picking up muck by hand and depositing it in the shaft bucket. Here again, some means of support for the equipment must be provided down in the shaft.

Recently Eimco Corp. mounted on crawler tracks a new, entirely redesigned underground, overshot loader, shown above. This machine, called the Eimco Model 630, has an overall length of 9 ft 4 in. with the loading bucket down, and 6 ft 3 in. with the loading bucket up. It is 5 ft 9 in. wide and weighs 5 tons. Power is provided by three air motors, one to operate the hoisting chain on the bucket and one attached to each crawler. The motor on each track permits excellent maneuverability—reversing one motor and moving ahead with the other turns the machine in its own length. The operator, standing on a running board, needs only three levers to air valves for complete control. With his left hand he operates two levers to control the tracks. (Not much leverage is required for this—one hand can control the forward, backward, and turning motions.) With his right hand he operates a lever controlling the air motor that hoists the bucket. A 1½ or 2-in. hose supplies air for the motors.

Actual operation of the machine requires two men, an operator and a tail hose man. In loading, a shaft bucket is placed near the shaft wall. The mucker travels forward, crowding its shovel into



A versatile underground machine—the Eimco 630.

the loose muck. When full, it turns, facing away from the empty shaft bucket, and then hoists the shovel up over the top to discharge the load into the bucket. The filled bucket then is raised to the shaft collar by the hoist on its return trip after it has lowered an empty bucket. The shovel on the mucker can be of various sizes to meet requirements. At the Intermountain Chemical Co. shaft at Green River, Wyo., a shovel of 9-cu ft capacity was used with 2½-cu yd buckets.

This arrangement makes a fairly simple operation. Everything is controlled from the bottom. The only signal required to the outside is for the mine hoist and the only line in the shaft is the mine hoist cable. No supporting structures or guides for the mucking operation are necessary.

The clamshell and overshot methods of loading vary considerably. With any method some time is consumed getting the muck pile to the desired shape when loading is first started. A fresh muck pile, just shot, is often very uneven. Clamshell loading into the shaft buckets can usually start immediately, although the first few buckets must be loaded slowly. Even with clamshells there is a certain skill in placing the shaft buckets for maximum speed in loading. Advantage should be taken of the natural swing or drift of the clamshell. If the shaft bucket is not placed properly with respect to the point where the full clamshell is picked from the muck pile, men on the tag line will have to work against the swinging weight of the clamshell. Consequently it is necessary to shape the pile for clamshell loading or for mucking with the Eimco 630. Experience has shown that 10 to 15 min may be needed to shape the muck pile for bucket loading. The required shape varies according to the size and cross section of the shaft. It has been determined that in a round shaft of 18 ft finished diam, a saucer-shaped muck pile is best.

Working across the shaft, the mucker keeps the center of the pile low and the edges high. With the edge high the mucker can back up against the rib to place the bucket or shovel lower and sloping downward. In this position the shovel starts digging sooner, and it is relatively easy to keep the center low and fill the shovel quickly. This method of operation perhaps would not apply in a different shape or size shaft. Experience is the only way to determine the optimum shape and method. This shaft is the first in which the mucker has been operated in such a restricted space, although it has been used in a much larger rectangular shaft.

Proper fragmentation is much more important with the Eimco 630 than with clamshell loading. When a clamshell is used, size of the muck is not too important provided, of course, the clamshell can handle it into the shaft bucket. Large pieces can slow up the operation because the clamshell tends to slide off big rocks and several passes may be necessary to pick them up. The Eimco 630 loads best when 80 to 90 pct of the fragments are 1 cu ft or less in size. If there are many large pieces the shovel skids over the top, and the pieces have to be rooted up into the shovel. This does not mean the 630 cannot load large rock. The machine can pick up pieces that are even too large to fit inside the shovel. But speed is the primary consideration in mucking and production rate is materially reduced by improper fragmentation. Extreme care should be maintained in drilling and blasting.

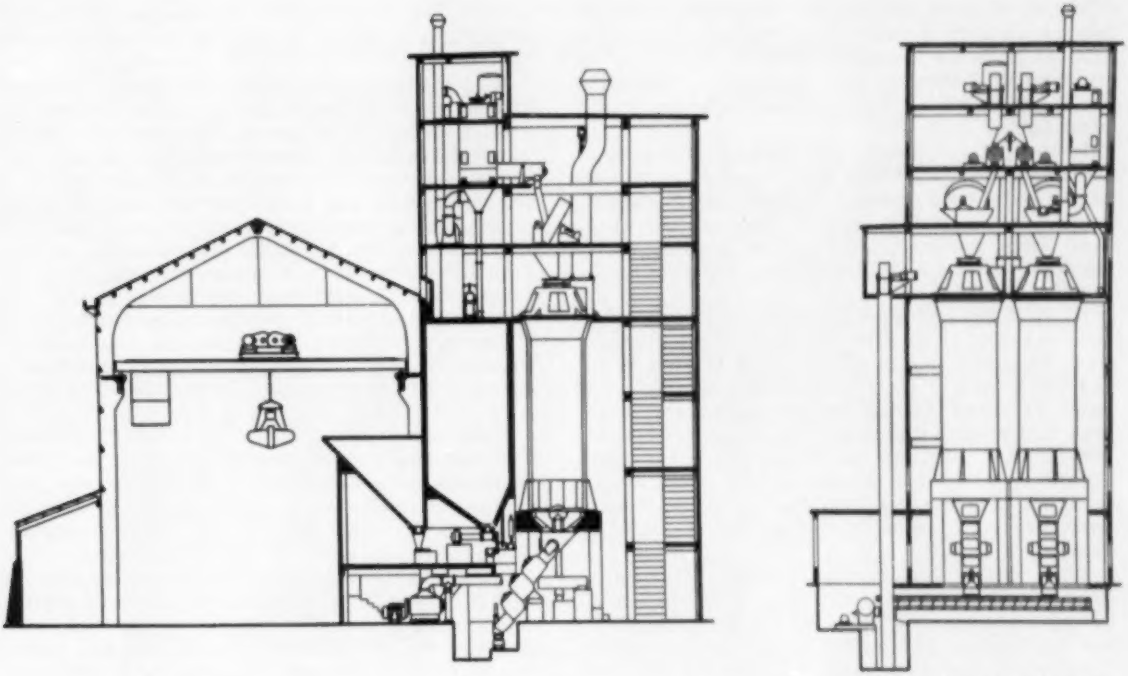
The volume of air required for the Eimco 630 is much greater than for an air-operated tugger on the closing line of a clamshell. Experience has shown that the 630 operates best in a shaft with about 750 cfm at a pressure of 115 psi. A tugger hoist requires about 200 cfm at 90 psi. Although some systems are operated electrically, air is better, especially in a wet or gaseous shaft. Air should not be too much of a problem on a shaft job, because drilling equipment usually requires more air than is necessary to operate the 630.

Manpower required for the 630 includes an operator, a hoseman, a helper for hooking and unhooking the buckets, and a signalman to the mine hoist. To use a clamshell with air-operated tugger, two tagline men are needed, a tugger operator, a signalman who signals both mine hoist and crane, and a helper to hook buckets and spell the tagline men. A set-up with the clamshell remotely controlled could be worked with an operator and two tagline men. With the bridge crane or similar systems an operator and two men to change buckets could make up the crew. Manpower varies according to the methods used and the policies of the contractor. Manpower on a shaft job is usually governed by the size of the drilling crew. At times a contractor uses what he believes to be a bare minimum of men for a given operation, but he may be sacrificing production and efficiency. For example, only two men are required to drift or work the taglines on a ½ or ¾-yd clamshell. But if these two men work for 8 hr, and hook and unhook the shaft bucket along with their tagline work, their efficiency and production rate will slow substantially towards the end of their shift. Today, with the rate of production and footage obtained in shaft work, it is impossible to have separate drilling crews and mucking crews, as was the case in former years. Today one crew of miners works each shift. This crew mucks, drills, sets forms, or places concrete—whatever happens to be scheduled on the shift. The number of men working per shift is based on the maximum number required for any one operation. Usually this is the drilling operation. Top labor is fairly inflexible. Each man usually is required to do a given job most of the time. This, together with union requirements for a job, makes it logical to keep miners assigned underground if they can be of any help in the operation. Sometimes an extra man can help a great deal. A good extra man hooking shaft buckets will increase the hoisting rate for an hour or a shift, even though tagline men could do the hooking.

Rate of production and footage sunk depends basically on the type of shaft, its size, and the kind of rock encountered, regardless of whether clamshell or crawler-mounted loader is being used. On the basis of experience gained sinking only one shaft with an Eimco 630 in comparison with many shafts sunk by clamshell loading, it would be unfair to quote production figures. Certainly they would not reflect an average for the Eimco 630. What can be said is purely a matter of judgment on the writer's part, and not a matter of record or based on fact. In the type of shaft sunk for Intermountain Chemical Co. at its Westvaco mine near Green River, Wyo., the Eimco would definitely outload a clamshell with an airtugger operating the closing line. This shaft is concrete-lined, 18 ft finished diam, providing a 20-ft diam in the rock section in which the mucker could operate. When conditions were optimum, that is, when muck loading on the shaft bucket was balanced with the time to make a round trip with the mine hoist, an average of 15 to 18 cu yd solid per hr could be loaded with the Eimco 630. It is probable that under the same conditions, 12 to 15 cu yd per hr could have been loaded by the clamshell method. Another advantage to be noted is that as soon as the mine hoist clears bottom with a loaded shaft bucket, the Eimco can start loading the empty bucket. The mine hoist can *highball* just as soon as he is given the outside signal. With any clamshell method, the rig cannot start loading until the shaft bucket is away from the danger of entanglement with supporting lines or structures. Time is gained, therefore, with the Eimco 630. It is fair to say that with all conditions equal the Eimco would be used in preference to a clamshell, if a shaft such as the one for Intermountain Chemical were to be sunk again.

Operating space, or lack of it, is a problem with clamshells and with the Eimco 630. Operating a clamshell in a shaft of less than 14 ft diam is hazardous. This diameter seems from experience to be the minimum. It has been proved that the Eimco will work easily in a shaft of 18 ft diam. It would be difficult to predict whether it could work a smaller shaft. It might be used in a shaft of 16 ft finished diam, which would provide an 18 ft diam working area. Before any statement can be made as to the minimum working area required for the mucker, the unit will have to be tested.

It is believed that the Eimco 630 is much safer to operate than any clamshell method thus far introduced. This is due primarily to the absence of entangling lines between the mine hoist cable and cables supporting the clamshell. In addition, there are no supporting structures overhead in the shaft that could be fouled with the mine hoist and shaft bucket. With a clamshell, there is always the hazard of slipping and falling beneath the clam as it is lowering into the muck pile. Also, the clam could fall over if too much slack occurs in the load line. These hazards are eliminated by the crawler-type mucker, which remains firmly on the muck pile, where the crew can watch every operation without worrying about anything overhead. This does not mean that safety precautions can be forgotten. It takes an alert and well coordinated team to muck productively with the Eimco. Each man must know what the other is about to do at all times. The Eimco 630 moves reasonably fast. The operator must be a capable man and keep the machine under close control. It is the safest policy to break in a crew slowly and train the men well. Let production follow.



Two-kiln de Roll vertical kiln plant

The De Roll Vertical Kiln

by H. Herbert Hughes

IN the years following World War II, L. de Roll S.A., Zürich, Switzerland, perfected its vertical kiln. De Roll first became interested in cement-manufacturing equipment in 1947, not only kilns but also the crushing, grinding, and handling equipment required for a complete cement plant. The parent company, Louis de Roll Iron Works, S.A., Gerlafingen, Switzerland, was founded in 1804. In its six plants, and with its associated companies, de Roll manufactures and installs equipment for a wide variety of industries—chemical and metallurgical operations, fats and oils plants, and municipal refuse incinerators and sewage disposal installations.

Vertical shaft kilns, of course, are as old as Portland cement itself. Their fuel economy has long been recognized, particularly in Europe where fuel costs are high. But the disadvantages of old-type shaft kilns have been the lack of consistent quality of the clinker, the small capacity, and the relatively high labor costs.

The basic problems faced by de Roll engineers in readapting the shaft kiln to modern cement manufacture, therefore, have been to provide means for maintaining a uniform charge to the kiln and proper and constant clinkering conditions as the charge moves downward through the kiln. The de Roll design directly reflects the solution of each of these fundamental problems to produce clinker from which cement of uniform quality can be made.

H. H. HUGHES is Vice President for Europe, Porter International Co.

By 1950 de Roll S.A., Zürich was ready to offer for commercial use its vertical kiln cement plants. The first installations in Switzerland and France started operating in 1951. Now 30 kilns are operating and 10 more are in various stages of construction. Locations of existing kilns, and those on order to be installed, are shown by countries in the accompanying table.

Table I. Location of de Roll Vertical Cement Kilns

Country	City or Town	Number of Kilns
British West Indies	Freeport	4*
England	Plymouth	2
France	Albi	1
France	Cruas	1
France	Crugey	1
France	Champagnole	1
France	Grenoble	2
France	Soulanges (Marne)	1
France	Xeuilley	1
Italy	Adria	1
Italy	Casale Monferratto	1
Italy	Este (Padova)	1
Israël	Haifa	4
Kenya	Mombasa	2†
Rhodesia	Colleen-Bawn	1
Spain	Alicante	1*
Spain	Bunol	3†
Spain	Malaga	2*
Spain	Valencia	2†
Switzerland	Brunnen	2
Switzerland	Darlingen	1
Switzerland	Liestal	1
Switzerland	Reuchenette	1
Switzerland	Siggenthal	1
Switzerland	Thayngen	2

* Under construction.
† One kiln under construction.

In the Western Hemisphere four kilns with an annual capacity of 1,200,000 bbl recently have been sold to Bahama Chemicals Ltd., a subsidiary of National Bulk Carriers Inc., New York. These kilns are to be erected at Freeport, Grand Bahama, British West Indies.

In the U.S., of course, standard ASTM specifications must be met. While it is true that no de Roll plant has yet been operated to serve the U.S. market, de Roll is certain that cement produced in vertical shaft kilns, designed and installed properly under the supervision of de Roll engineers, will meet ASTM specifications.

De Roll vertical kilns are built in two standard sizes: 1) the 75-ton kiln, 6 ft diam, with output of 75 to 85 metric tons of clinker in 24 hr, and 2) the 150-ton kiln, 8 ft diam, with output of 150 to 170 tons. Principal interest centers around the larger unit, and present discussion will be confined to the 150-ton kiln, although the 75-ton plant may prove extremely interesting for special jobs where the installation may be designed for maximum ease of recoverability for moving such a plant to a new location.

Plant design customarily provides for installation of two 150-ton kilns; thus, a 300-ton plant (600,000 bbl per year) may be considered a standard installation. The second kiln, of course, may be installed at any time after the original first one, along with additional equipment necessary for doubling capacity for raw meal and finished product. Actual original construction costs to provide for such expansion are somewhat higher than for a one-kiln 150-ton plant, but far less than to add later a separate 150-ton installation.

Raw Materials Requirements

Certain operations in a vertical-kiln plant are virtually identical with those in a rotary plant. Quarrying, raw meal preparation, and cement finishing are about the same in any cement plant irrespective of type of kiln. This description will start, therefore, with raw meal and fuel ready to be charged and will end with clinker ready for the finishing department.

Raw Meal: Broadly, chemical composition of the raw meal for the shaft kiln is the same as for a rotary kiln. It is, of course, essential to have uniform distribution of the various constituents, uniform fineness, and an homogeneous condition. For proper clinkering in a shaft kiln, however, it is important too that the content of iron and alumina should be high, both actually and in proportion to silica. It may, in some instances, be necessary to add alumina in the form of clay, or even as bauxite, to increase the alumina content, and under certain conditions, it may be advisable to add roasted pyrites or iron ore to increase the iron content. The most favorable relationship for shaft kiln raw material is the following:

Lime standard

$$\frac{100\,C_aO}{2.8\,SiO_2 + 1.1\,Al_2O_3 + 0.7\,Fe_2O_3} = 95 \text{ to } 100\,.$$

$$\text{Silica ratio } \frac{SiO_2}{Al_2O_3 + Fe_2O_3} = 1.8 \text{ to } 2.5\,.$$

$$\text{Alumina ratio } \frac{Al_2O_3}{Fe_2O_3} = 1.2 \text{ to } 1.8\,.$$

These figures are not absolute, however, as variations from these standards are giving excellent results in some shaft kiln installations But it is important that once a given composition has proved satisfactory it should be kept as uniform as possible, especially in regard to the lime standard.

As for the fineness of the raw meal, increased fineness gives improved quality of the finished cement. Obviously, however, the cost of electric power for grinding demands selection of some optimum standards, and for the shaft kiln this limit lies between 10 and 15 pct retained on a 170-mesh screen. Some variation in this may be possible. Raw material that is readily burned, easily disintegrated, very uniform, homogeneous, and of marly nature may be somewhat less finely ground and give excellent results. On the other hand, poorly clinkering material of irregular composition, blended from constituents having wide extremes in chemical analysis, will require finer grinding to insure good clinker.

Fuel: All de Roll kilns in use to date burn solid fuel, and fuel consumption is relatively low. Heat requirements normally are slightly less than 1800 Btu per lb of clinker, or about 650,000 Btu per bbl of 376 lb. Typically 58 lb of anthracite buckwheat N°4 or N°5 are needed per barrel of cement.

Either coke or anthracite is required because the fuel must contain as small a proportion of volatile constituents as possible; otherwise such high volatile fractions are distilled in the drying and calcining zone of the kiln and thus are not available for combustion in the clinkering zone. Also, the calorific value should be as high as obtainable, preferably 11,700 to 12,600 Btu per lb. The minimum is about 9,900 Btu per lb in a grade of coke having 22 pct ash and 10 pct water content. Lower calorific values affect the kiln output and the quality of the clinker, for there is danger of under burning unless the required clinkering temperature is maintained uniformly throughout the burning zone of the kiln. The nature of the ash too is important. Its analysis is allowed for as part of the charge to be converted to clinker. Samples of limestone, clay, and fuel must be approved by de Roll engineers and chemists before a kiln is delivered.

To be completely satisfactory the fuel should comply with the following specifications, although fuel varying from these standards has been used satisfactorily in some installations:

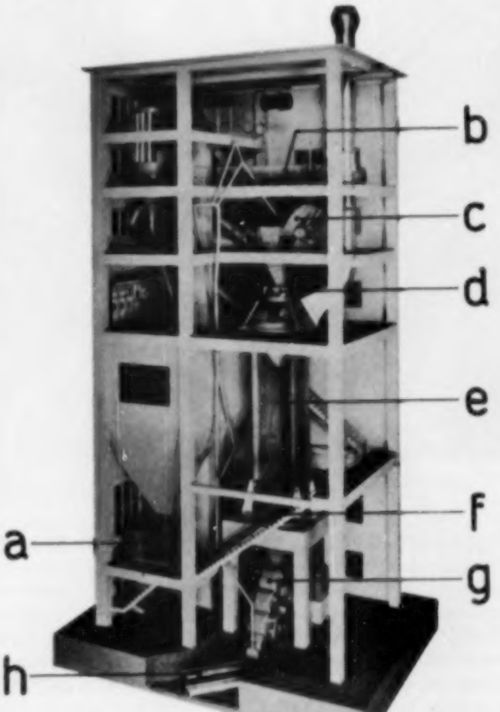
Item	Anthracite	Coke
Volatiles, pct	6 to 8	2 to 4
Ash, pct	6 to 15 (approx.)	6 to 15 (approx.)
Water, pct	4 to 6	4 to 10
Calorific value, Btu per lb	11,700 to 12,600	11,700 to 12,600

Particle size likewise is important, although coke or anthracite screenings of maximum ⅜ in. generally may be used as delivered without drying or crushing. A high proportion of fines, under 3/64 in., should be avoided, as they increase the quantity of carbon monoxide in the stack gases. Also, an excess proportion of particles above ¼ in. is undesirable, as these large pieces burn longer, increasing depth of the burning zone, which results in too high a clinker temperature at the discharge gates. Both an excess of fines and an excess of coarse particles have an unfavorable effect on fuel consumption.

Two new developments in fuel utilization promise to expand widely the potential application of the shaft kiln. The first is that petroleum coke, although not yet used regularly in commercial production of cement in a de Roll shaft kiln, has been tested on a commercial scale, and the results of such tests show that it is eminently satisfactory as

Detailed Description of De Roll Kiln Installation

Full details of techniques relating to manufacture and operation of important features of the kiln are zealously guarded by de Roll and not available for publication. Nevertheless, the description shows adequately a typical installation. All comments refer to a standard 150-ton kiln; the optimum plant is either an installation of two such kilns, or again doubling to four kilns.



As contrasted with the unsatisfactory performance of early shaft kilns, operation of the de Roll kiln is as follows:

a) Raw meal and fuel are charged in exact proportions by weighing machines.

b) Helical mixer blends the raw meal and fuel into a charge of uniform constituency.

c) Charge passes to a nodulizing disk where, with the addition of about 12 to 14 pct water, the charge is formed into nodules, mostly ranging in diameter from about ⅜ to ¾ in. The few larger nodules that may form are not detrimental.

d) Adjustable rotary hopper properly charges the nodules into the kiln, maintaining a uniform charge over the entire diameter.

e) Nodules pass down through the kiln proper. Observations on kilns in use show that most of the nodules keep their shape until actual discharge from the kiln. This means retention of uniform permeability of the charge and consequently uniform gas distribution over the whole section of the kiln, with the clinkering zone remaining constantly at the same general level.

f) Rotary grate breaks up caked clinker. Although the nodules generally keep their shape, they tend to fuse into irregular clinker masses that are broken up by the rotary grate. Speed of the rotary grate can be varied, thus regulating the quantity of clinker extracted and by so doing controlling the burning process.

g) Discharge chute comprises three discharge gates fitted with airlocks to prevent escape of combustion air, so that continuous discharge of clinker has no effect on burning conditions.

h) Clinker is dropped on shaker conveyor.

Simple jaw crusher, not shown, reduces caked clinker to maximum 1¼ in. ready for finishing department. Blower forces combustion air through rotary grate in adequate volume to maintain proper conditions in burning zone. Introduction of such air at the bottom of the kiln serves also to cool the clinker between burning zone and discharge gates.

a fuel. The second, the introduction by de Roll, in the near future, of an oil-fired (or gas-fired) shaft kiln will be discussed after the detailed description of the solid fuel kiln. In this sequence the modifications will be more clearly understood.

Weighing and Mixing: Referring to the schematic diagram, the first operation unique to the shaft kiln is the weighing and mixing of raw meal and solid fuel in proper proportions. In present installations this is done by automatic batch weighers, one for raw meal and one for coal, the intermittent batches passing directly to a helical mixer to distribute the fuel uniformly throughout the charge.

This system, however, is being superseded by electronic-control continuous weighing machines, developed by de Roll expressly for the shaft kiln, and already thoroughly tested in one installation. With continuous feed in exact proportions at all times, the two weighing machines discharge into a worm conveyor (either directly or after elevators, depending on location of weighing machines) that delivers the proper mixture to the nodulizer. The helical mixer is no longer required.

Exact proportions of raw meal and fuel must be determined for each blend of raw meal and grade of fuel. A common example is 84 pct raw meal and 16 pct fuel.

The new continuous weighing machines cost a little more than the old system but the opportunity for closer control makes far better burning conditions, which are reflected in increased output with a measurable saving in fuel consumption. All recent orders for shaft kilns include the improved weighing system. In present installations horsepower for the raw meal weigher is 1.5 hp, for fuel 0.3 hp, and for the worm conveyor 3 hp.

Nodulizer: Although no one part of an integrated manufacturing process can be singled out as the key to such an operation, the nodulizer that prepares the physical form of the charge to the shaft kiln is certainly of major significance in a de Roll installation.

Proper clinkering in a shaft kiln requires that the charge be permeable to the combustion gases. This nodulizing of the raw meal and fuel mixture, not only to control closely the shape, size, constit-

uency, and fuel distribution of each nodule, but also to provide permeability, makes possible the uniformity of the clinker, and consequently that of the finished cement. This system of kiln feed preparation, combined with the combustion and clinker discharge controls of the de Roll shaft kiln, produces clinker with characteristics comparable to that produced in any other type of kiln.

The nodulizing disk consists essentially of a large inclined pan 8 ft 2 in. diam and 2 ft deep, rotated by a direct V-belt drive from a motor with installed capacity of 10 hp. Total weight of the nodulizer is about 3.5 tons and the actual charge in the pan about 3 tons.

Speed of rotation and angle of inclination are constant, being adjusted to fit the characteristics of the material to be nodulized. For flexibility of operation, however, to vary the size or density of the nodules, two controls are possible—rate of feed charged and rate of water flow. Nodules are discharged from the rotating disk by the centrifuge effect.

Capacity of the nodulizer is roughly 11 tons of nodules per hr, the rate necessary to produce 150 tons of clinker in 24 hr on the basis of 1.56 tons of raw meal per ton of clinker and about a 5.25 to 1 relationship between raw meal and fuel. The device is designed to produce 92 pct of nodules in the range of 5/16 in. to ⅝ in. with 8 pct over ⅝ in. and none smaller than 5/16 in. In actual operating practice somewhat larger nodules may be charged to the kiln, and tendency of some material to build up an occasional quite large nodule before it is finally discharged is of no consequence, for a few such large lumps make no differnce in the kiln.

Caking on the sides and bottom of the pan varies with the character of the feed, but normally it is no great problem. What little there is usually can be removed by improvised hand tools. A simple mechanical scraping device designed by de Roll may be attached to the pan if actually needed.

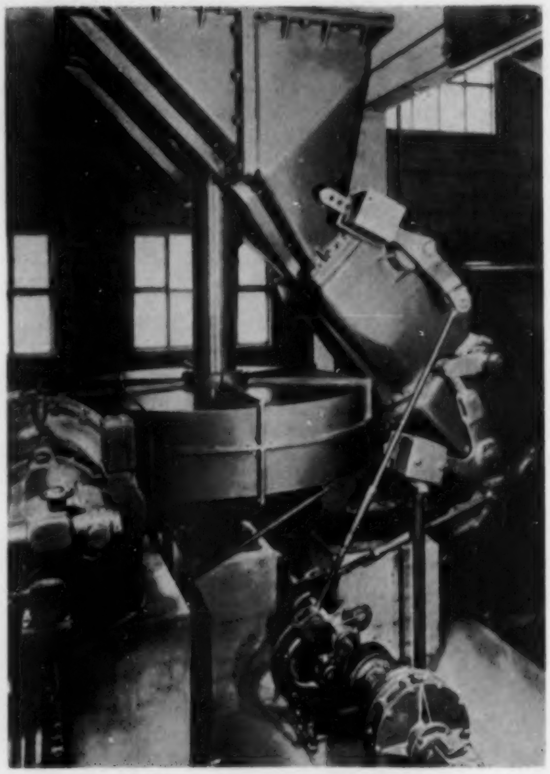
The nodulizing disk finds applications elsewhere than for preparation of proper feed for the de Roll vertical cement kiln. Some European cement manufacturers have switched recently to this dry method of feeding nodules into rotary kilns and have effected substantial savings in fuel consumption. Also interesting is its use for fertilizers, fine ores, and various products of the chemical industry.

Charging Hopper: Nodules are fed into the kiln through a charging hopper of special design. Its principal feature is a revolving chute that discharges nodules uniformly over the full diameter of the kiln. Inclination of the chute and its speed of rotation may be varied to provide proper charging. The mechanism is driven by a 2.5-hp motor.

Kiln: The kiln proper is a vertical cylinder made of steel plate, about 33 ft high and 9 ft 4 in. OD. When lined with proper refractories ID is about 8 ft. Except for the grate at the bottom, there are no parts of any kind, moving or stationary, throughout the entire height.

Over the top of the kiln, above the burner platform, is a sheet metal hood in which the charging chute rotates and from which the stack gases are drawn. For visual inspection of the top of the charge this hood is equipped with doors, through which long steel rods are occasionally plunged downward manually to make sure that the charge is moving properly through the clinkering zone.

Average temperature at the top of the charge is around 100°C (212°F). The top 5 ft of the kiln constitutes the drying and calcining zone, and the 10 ft below the clinkering or burning zone. Close control in this zone, both of temperature at 1450°C (2632°F) and rate of movement of the charge downward through the zone, are imperative for producing good clinker. The bottom 18 ft of the kiln is the cooling zone, discharge temperature about 250°C (482°F).



Driving mechanism for the rotary grate and discharge grates.

Much research has gone into the problem of finding the best possible kiln lining, and with good results, but details are not available for publication. For the top drying and calcining zone, the lining must be unaffected by humidity and resistant to wide change of temperature. In the clinkering zone, specifications call for a refractory that will take up to 1850°C (3352°F). In the cooling zone, resistance to abrasion is the main problem along with specification for 1710°C (3100°F).

The task of relining a de Roll shaft kiln is relatively simple. In the first place, only the clinkering zone of about 10 ft has to be relined on a definite schedule. Of the 30 kilns in operation, none has yet been relined in top and bottom zones, although some have now been operating for more than five years. For the clinkering zone, relining is necessary about once a year, although in actual practice kilns are being operated as long as 400 to 500 days before relining. A typical relining job requires a nine-day shutdown—three for cooling, three for relining, and three for reheating. There has been no problem of material adhering to the sides of the kiln to complicate the relining job.

A unique feature of the de Roll kiln is that it may be shut down for 24 to 48 hr, with the fire

banked, and production can be resumed simply by again starting the machinery in operation. Actually, however, such operating practice as a six-day week is not recommended, for it does take 3 to 4 hr to bring the kiln back up to proper heat after it has been shut down, and thus production is lost for about half a shift. For emergency repairs to equipment, for special holidays, or for any other such reason, the ease of shutting down is distinctly advantageous. Kilns have been down for as long as one week without losing fire but, after so long a shutdown, priming with extra fuel is necessary to start up again.

Rotary Grate: For continuous discharging of clinker, a rotary grate is mounted at the bottom of the kiln. It consists essentially of a base body steel casting that supports grate plates with cast-on scrapers covered by a welded protective lining of high abrasion-resisting material. The grate itself, exclusive of the drive shaft and driving mechanism, is shaped roughly like half a grapefruit, higher in the center, 8 ft 2 in. diam and 3 ft 4 in. thick at its central point.

The grate revolves horizontally very slowly, from one to six revolutions per hr, depending upon individual kiln conditions. No grate in any installation has yet been replaced, but the special hard abrasion-resistant tips on the teeth of the scrapers have to be renewed by welding every two or three years. The grate requires a special three-phase motor of the shunt collector type for continuous speed variation. Installed capacity is 7.5 hp.

Discharge Gates: The three-compartment discharge gates are of special design to prevent loss of combustion air from the kiln as clinker is discharged. Each gate is about 3 ft 4 in. high by 3 ft 4 in. wide by 4 ft deep, so the complete discharge-gate mechanism is about 10 ft high. Operation of the gates is mechanical, timed for 3.3 cycles per min. At each gate the clinker passes downward from one to the other on a staggered basis so that no more than one gate ever is open at one time.

The gates must be built for heavy duty to resist the abrasive action of the clinker. They discharge 6.3 to 6.5 tons of clinker per hour, thus dropping 60 to 70 lb every 18 sec. In particular, the frames that are hinged at the top, and open and close to allow passage of the clinker, take a lot of punishment. Even so, made of special cast steel, they require replacing only about once a year. As they are designed for ease of maintenance, this job can be done in less than half a day. Clinker is discharged directly on a shaker conveyor which takes it to a small crusher—maximum 1¼ in.—and then to clinker storage. Plants in Europe now operating both rotary kilns and de Roll shaft kilns at the same location combine their clinker from both types of kilns indiscriminately in clinker storage.

Blower: Each shaft kiln requires blower capacity of about 5.000 cu ft of air per min. This quantity may vary somewhat with the characteristics of the raw meal and the nature of the fuel. Also, for flexibility of operation, and as a safeguard against shutdown, there may be an advantage in designing for two or three smaller blowers rather than one large one. In either case, the blower installation will require 80 to 100 hp, installed capacity. No heat exchanger has been used on the solid-fuel kiln.

Electrical Installations: A multiple-panel switchboard with controls and recorders for all operations is installed with each kiln. Among such controls are those for the weighing machines, kiln feeding, combustion air, and kiln extraction, as well as for the silos ahead of the kiln and clinker transport following discharge. Recorders measure quantity and pressure of combustion air, temperature of exit gases, and speed of rotary-grate motor.

Charge being fed in the form of nodules.

A measuring diaphragm for combustion air, a thermoelectric pyrometer, a solenoid valve for the water supply to the nodulizer, and various remote control push-button boxes are mounted separately as convenient.

Failure of any part of the plant immediately shows up on the appropriate panel, and all other operations relating to the stoppage are halted automatically.

Electric current consumption for the entire kiln department is 12 to 14 kw-hr per ton of clinker, or 2 to 2.4 kw-hr per barrel.

Dust Collecting System: There is nothing unique about dust collection in a shaft-kiln plant except that the system, of course, has to be designed to fit the problem. Otherwise, it is the same as in any cement plant.

Employes

In European practice, no fewer than two men per shift are required for a de Roll shaft kiln installation of one or two kilns. Direct labor in the kiln department averages about 0.04 man-hr per bbl of cement.

The first man, of no particular skill, works most of the time on the nodulizer floor, keeping a visual check on the nodulizer to see that it is functioning to produce nodules of proper size and consistency. He also cleans the entire kiln building and looks after all manually lubricated bearings. The second job must be filled by someone of greater skill. This man, the burner, works all the time on the burner floor at the top of the kiln. He must maintain proper clinkering temperature in the kiln by control-

ling combustion air, and must regulate, too, the proper rate of charging nodules and discharging clinker. By experience he learns to *feel* the kiln charge by plunging long steel rods down into the clinkering zone of the kiln and, using only this one manual aid along with the various controls under his jurisdiction, he is responsible for both quality and quantity of clinker.

Although the human element is important to proper functioning of the kiln installation, automatic control devices are employed wherever possible, particularly in case of trouble. If anything goes wrong, all other related operations are stopped automatically until the trouble is corrected.

Each shift foreman, of course, checks periodically on all phases of the kiln operation during his rounds of the plant.

Maintenance

Already mentioned is the need for rewelding, every two or three years, the abrasion-resistant tips on the teeth of the rotary grate scrapers and for replacing about once a year the frames that open and close to permit discharge of the clinker through the specially designed gates. In addition, of course, a continuous maintenance program comparable to that in any installation of heavy industrial equipment must be maintained.

The bank of spare parts to be carried at any kiln installation varies widely, however, depending largely where the kiln is located with respect to access to de Roll factories. For example, for a plant in Africa or any other remote location, spares may amount in cost price to as much as 10 to 12 pct of the original equipment itself. This is for a one-kiln installation. For two kilns, or even for three or four, the total bank of parts need be only nominally higher. The most expensive item commonly carried is a set of grate plates. No kiln in operation has yet needed new grate plates, but most plant operators are unwilling to risk not having a spare set available. Whether the plant comprises one kiln or four kilns, one spare set of grate plates may be considered as adequate protection.

Most of the inventory consists of bearings, gears, rings, bolts, comparable relatively low-cost items required for day-to-day preventive maintenance and for replacement of parts likely to cause unexpected shutdowns. Maintenance has not proved a major problem in de Roll vertical kiln installations.

Economics

To mention briefly the economic advantages of the de Roll vertical kiln, perhaps the most significant feature is that plants with an annual capacity of 1,200,000 bbl, 600,000 bbl, 300,000 bbl, or, with the use of one 75-ton kiln, even 150,000 bbl are competitive. In fact, capital costs per barrel of annual capacity are substantially lower than for other types of kilns.

In the U. S. emphasis has been on expansion of capacity in units far larger than the average de Roll plant. However, for a plant strategically located to serve a particular metropolitan market area, perhaps even a single central-mix concrete plant, or for a plant in a remote location to be amortized during a specific dam project or some other heavy construction job, de Roll installations are attracting increasing attention.

Engineers studying the problem of designing a de Roll vertical kiln plant for maximum ease and economy in relocating the plant after its need in its original location has ended are confident that 80 to 90 pct of the first cost can be recovered for relocation elsewhere. Packaged power gas-turbine generator units of 5000 and 10,000 kw, especially designed for ease of relocation, already are in use for other purposes in remote areas. Instead of mass concrete foundations, these units rest on structural steel skid-type bedplates, which in turn are leveled by jack screws on plain concrete slabs poured at grade. Applying the same principle to the rest of the plant, it would be designed with steel silos and bins, steel frame bolted construction, air conveyor systems, service lines above grade, and other built-in features to minimize relocation expense.

This ease of relocation should not be unduly emphasized. Operating costs to produce a barrel of cement in a de Roll installation compare favorably with other types of plants. Thus any potential plant location in which a supply of proper raw materials is combined with an adequate market to absorb the output of the plant is worthy of consideration. Especially this is true if the location is such that substantial freight savings over existing sources of supply of cement are possible.

Oil-Fired Kiln

Virtually ready for announcement by de Roll is a modification of the present shaft kiln to burn either oil or gas as fuel. The most essential new feature of the design is an oil-burning (or gas-burning) device surrounding the kiln at the clinkering zone from which the hot gases at proper temperature are drawn through the charge as it passes down through the kiln, which probably will be oblong in cross section rather than round.

Another important feature is a heat exchanger with both the cooling air from the discharge end of the kiln and part of the stack gases used to preheat the combustion air. Fuel efficiency, of course, is directly dependent upon the effectiveness of the heat exchanger. More air is needed than for the solid fuel kiln, so a different and bigger blower installation is required, and total power consumption will be a little higher.

On the other hand, charging is simplified compared to the solid fuel kiln, with no coal handling and no mixing equipment necessary. Otherwise, there is no change in operation—the nodulizer, charging hopper, rotary grate, and discharge gates function in exactly the same manner as with a kiln burning coal or coke. The same employes are required, although the burner has somewhat different duties.

The oil-fired shaft kiln may be expected to be more expensive than the present solid-fuel unit. There is every indication, however, that clinker capacity also may be increased, so on a comparative basis of capital cost per barrel of annual capacity there may well be a completely favorable relationship.

Patents have been applied for. Further details on the oil-fired shaft kiln are not yet available but probably will be revealed by de Roll in the near future.

Actual introduction of the oil-fired (or gas-fired) kiln will intensify mounting U.S. interest in the de Roll vertical kiln.

Coal Preparation at The Jones & Laughlin Vesta Mines

by J. A. Glunt and J. R. Dawson

VESTA No. 4 and 5 mines supply most of the high volatile coal required for Jones & Laughlin's by-product coke plants. Until 1944 all coal produced in these mines was loaded by hand. Pressure to mechanize at Vesta was brought on by the manpower shortage in World War II, and in 1943 a decision was made to change to mechanical loading. It was the opinion of those concerned that by use of selective mining the coal would be clean enough to be processed by the existing preparation plants. At that time coal cleaning at Jones & Laughlin consisted of hand picking the larger sizes at the mine tipples and wet washing at plants located at the Pittsburgh and Aliquippa Works.

A typical section of the Pittsburgh seam as it occurs at Vesta is shown in Fig. 1. The 14 to 16 in. of drawslate immediately over the coal is too soft to be used for roof and must be taken down, leaving the rooster coal just above for roof. In selective mining an attempt is made to mine the coal under the drawslate first and then load out the drawslate separately as refuse. In full seam mining slate and coal are shot down at the same time and loaded out.

Mechanical loading at Vesta created new problems. It was found that selective mining, even when carefully applied, increased the refuse in the raw coal to such an extent that the washers at the mills were unable to handle it. Large tonnages of refuse were being transported, a reject disposal problem was created, and good metallurgical coal was being loaded out and lost. In addition, the selective method handicapped operation of the loading machines. Fig. 2 shows how the ash content of the raw coal increased as mechanization proceeded at No. 5 mine.

J. A. GLUNT is Ass't Chief Metallurgist, Cleveland Works, and J. R. DAWSON Chief Chemist, Vesta-Shannopin Coal Div., Jones & Laughlin Steel Corp.

TP 4380F. Manuscript, Sept. 26, 1955.

New Preparation Plant: After an exhaustive study of this situation by J & L management, is was decided that a preparation plant should be constructed at the mines, capable of handling full seam mined coal and large enough to process full production of both No. 4 and No. 5 mines. It was decided that from the standpoint of inside operation and haulage the new plant should be located as near as possible to the old No. 5 tipple on the Monongahela River near Vestaburg.

Full seam mining at Vesta could be expected to produce a tremendous volume of refuse along with the coal, since 14 to 16 in. of slate would be mined with an average 78 in. of coal. Disposal of this refuse required serious consideration in selecting a plant site. The only area near Vestaburg large enough for disposal of the plant refuse was across the Monongahela River at LaBelle. Ample space was available here, and construction at this location would not interfere with operation at the No. 5 tipple.

It had been noted over the years that the sulfur content of the coal produced at Vesta was gradually increasing, and there was every indication that this increase could be expected to continue as the mines developed further from the river. It was decided to construct a three-part separation plant with complete blending facilities to produce both metallurgical and steam coal. In this system an attempt is made to reduce the sulfur in the metallurgical coal in a secondary cleaning process.

Construction of the new preparation plant got under way early in 1947 and continued until the spring of 1950. The plant was designed to handle 2400 tph of raw coal containing 30 to 35 pct refuse. Cost of outside buildings and equipment was about $14 million. Total cost of the plant, together with improvements and changes made at this time, was approximately $21 million.

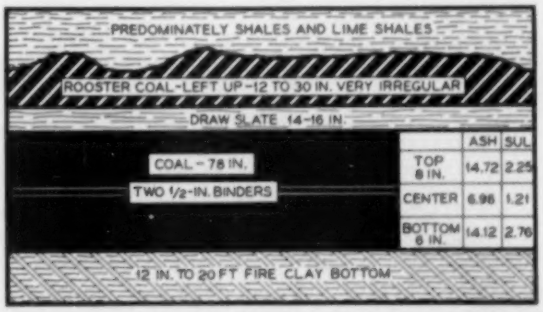


Fig. 1—Vertical section of Pittsburgh coal seam occurring at Vesta. Drawslate overlying coal creates problem in mining.

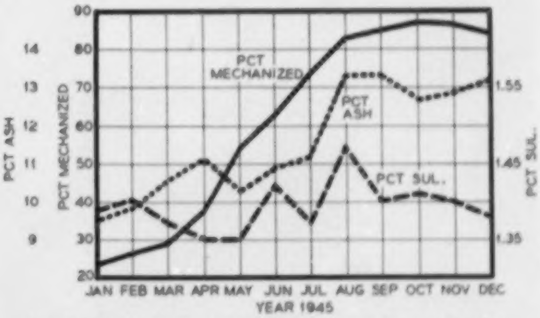


Fig. 2—Mechanization of loading operation increased refuse in raw coal delivered to mill washers.

The Transactions papers appearing in Mining Engineering during 1956 will subsequently appear in AIME Transactions Volume 205, and may be permanently referenced to that volume.

Fig. 3—Refuse stacker. Each day of plant operation 8000 to 9000 tons of refuse are added to this pile.

Fig. 4—The rotary dumps handle nine mine cars at one time.

Operational Data: Raw coal from both the No. 4 and No. 5 mines enters an underground chamber in mine cars. Nine cars at a time are handled by two rotary dumps, shown in Fig. 4. Coal is discharged into a common hopper with a conveyor bottom, from which it is fed over an 8-in. bar screen. Screen oversize is crushed to —8-in., and screen undersize and the crusher product are recombined and loaded onto the 60-in. conveyor leading to the plant. The belt structure crosses the Monongahela River on a 1000-ft suspension bridge. On entering the plant the coal is evenly distributed by means of a traveling belt tripper over the 16 sections of the raw coal bin. The coal is fed from this bin by vibrating feeders to 16 double-deck vibrating screens. These screens effect a separation at ¼ in., the +¼-in. material going to the coarse coal cleaning plant and the —¼-in. to the fine coal plant.

From the vibrating screens the 8x¼-in. raw coal is sent to eight primary heavy media vessels. These units are rated at 250 tph so that seven vessels can handle the entire production. Operating at 1.55 sp gr, these vessels separate the heavy refuse, which is immediately sent to the refuse disposal system shown in Figs. 3 and 6. Float coal from these primary vessels is conveyed to blending bins ahead of the secondary heavy media vessels.

The six secondary vessels, each having a capacity of 175 tph, are operated at an effective 1.35 sp gr. The float product is metallurgical coal and the sink an intermediate product that is then screened at 1 in. The 1x0-in. is sent to steam coal blending bins while the oversize is crushed to —1-in. and returned to the raw coal feed. The 8x¼-in. 1.35 float coal is sent to the metallurgical coal blending bins.

The —¼-in. material from the raw coal vibrating screens is conveyed to blending bins and then to four Rheolaveur launders, where it is washed at 1.55 sp gr. These launders are augmented by 12 Deister concentrating tables that are used to treat 1/8x0-in.

Fig. 5—Three Dorr thickeners are in foreground. Tall building at rear contains blending bins.

Fig. 6—Movable section of refuse conveyor between head house and stacker.

Fig. 7—Bank of nine carpenter driers. A similar bank to the right cannot be seen. On extreme left are eight primary heavy media vessels.

Specific Gravity of Materials Present in Raw Coal

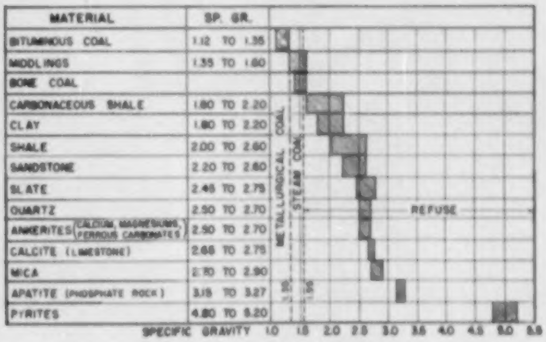


Fig. 8—Separation at 1.55 removes heavy impurities. Separation at 1.35 removes middlings and bone coal.

launder middlings. Dewatering of the clean fine coal is accomplished by means of four boot elevators and 18 continuous centrifugal driers, Fig. 7. Clean fine coal is sent to the metallurgical blending bin.

As in all wet preparation plants, one of the major problems at Vesta is clarification of the plant water. This is especially true now that state laws forbid discharging contaminated water into the streams. Equipment at Vesta for this purpose consists of three Dorr thickeners, two 150 ft diam and one 85 ft diam, see Fig. 5, and six disk-type vacuum filters. The filter cake is recovered as metallurgical coal. Some 14-in. cyclones have been added as auxiliary equipment since the plant was put into operation.

The Heavy Media Process: The heavy media process is used to clean the +¼-in. portion of the plant feed. The primary vessels, operating at 1.55 sp gr, effect a separation between coal and refuse, while the secondary vessels, at 1.35 sp gr, effect a separation of low ash-low sulfur coal and high ash-high sulfur middlings and bone coal, as indicated in Fig. 8.

Dense media processes employ pseudo liquids or liquids heavier than water to separate materials on the basis of specific gravity differences. Pseudo liquids are suspensions of solids in a fluid, the mixture having characteristics and properties approaching those of a true liquid. At Vesta the medium employed is a suspension of finely ground magnetite in water. Water currents provide enough agitation of the bath to hold the magnetite in suspension. At the present time flue dust from the Benson Mines sintering plant is being used. Screen analysis of this material is 95 pct —100 mesh and 50 pct —325 mesh. Loss of magnetite is about a half a pound per ton of raw coal feed to the vessels. A thickener and magnetic separator is provided along with each vessel for recovery of the magnetite.

Operation of the heavy media vessels such as are in use at Vesta is very simple. When the feed enters the vessel, the material heavier than the effective specific gravity of the bath sinks to an elevator, from which it is discharged. The lighter material, along with a certain portion of the medium, overflows a wier at the discharge end of the vessel. Both the float and sink products are discharged onto separate drain and rinse screens provided for recovery of the medium.

The Rheolaveur Launder: Since the Rheolaveur launder had been successful in operation at the J & L Hazelwood washer on fine Vesta coal, this type of equipment was selected for cleaning the —¼-in. coal at the new plant. The installation consists of four parallel units containing five launders each. In each unit the launders are arranged one below the other with *A* launder on top and *E* launder at the lowest level. These launders are essentially long sloping troughs with discharge boxes located at intervals along the trough bottoms. These boxes discharge heavy material to the launder at the next lower level. Rate of discharge is controlled by the size of the orifice in the bottom of the box and by upward currents of water through the boxes.

A three-part separation is effected in the launders. Overflow from launders *A*, *B*, and *C* is clean coal. Overflow from launders *D* and *E* is middling product. These middlings are screened at ⅛-in., the + ⅛-in. material being recirculated to the launder feed and the — ⅛-in. going to Deister tables for further cleaning. Underflow from a portion of *D* and all that from *E* launders goes directly to refuse.

Launders have a capacity of roughly 4 tph per inch of width of *A* launder. At Vesta these launders are 46 in. wide and have a capacity of 180 tph. The fine coal plant, consequently, has a feed capacity of 720 tph.

Liquid-Solid Cyclones

Inadequate Dewatering Facilities: Getting a new plant into operation is usually accompanied by unforeseen difficulties. The start-up at Vesta was no exception, for trouble was immediately encountered in the fine coal plant. Failure of the fine coal plant to operate as planned was due primarily to inadequate facilities for dewatering the washed fine coal.

Initial dewatering of this coal was to be accomplished by two dewatering wheels operating in parallel ahead of continuous centrifugal driers. Under actual operating conditions, the dewatering wheels were found to have only a fraction of the necessary capacity. It was for this reason that the first cyclones were installed at the Vesta preparation plant in August 1950.

Cyclones as thickeners and classifiers of coal and refuse slurries had been reported in operation at other coal preparation plants previous to installation at Vesta. Limited experience had also been gained in 1949 at the Aliquippa coal washer by the Coal and Coke Research Div. While still in the developmental stage the liquid-solid cyclone had shown definite advantages over conventional thickeners:

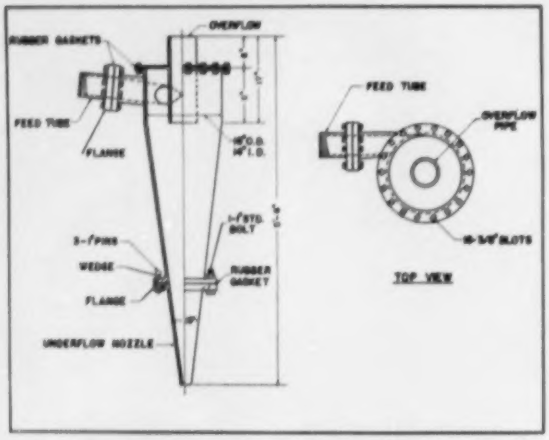


Fig. 9—Half section of 14-in. cyclone used at Vesta. Wall is ½-in. cast iron. These cyclones are cast, machined, and assembled at the J & L Pittsburgh works.

1) high capacity, 2) low initial cost, 3) no moving parts, 4) comparable efficiency, and 5) minimum space requirements.

As a general rule, the diameter of the cyclone selected for a given application depends on size consist of the feed particles, mesh size of separation desired, and volume of feed slurry to be processed. The large cyclones can handle coarser feeds and have the advantage of high capacity but on the other hand will allow larger particle sizes to report to overflow. The 14-in. cyclone, see Fig. 9, is now used exclusively at Vesta.

Use of Cyclones Prior to Boot-Elevator Operation: Early in November 1950 a decision was made by management to discard the dewatering wheels and install four boot elevators for primary dewatering of the fine clean coal. At the same time work was started on installation of twenty 14-in. cyclones and two 6x14-ft dewatering vibrating screens in order to get the plant operating on at least a temporary basis. (Fig. 10 is a flow diagram of this installation.) This permitted near capacity operation of the fine coal plant from Feb. 15, 1951, until completion of the fine coal boots and elevators, April 20, 1951. During this period the fine coal plant operated at an average feed rate in excess of 500 tph, and 310,000 tons were processed.

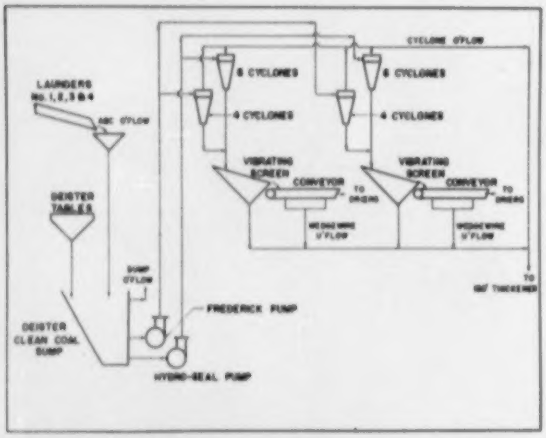


Fig. 10—This installation permitted operation of all four launders until completion of boot elevators.

Operating Costs: An estimate was made for the cost of primary dewatering of the fine coal using cyclones and vibrating screens for the month of March 1951. The cost was found to be $0.0425 per ton of raw coal fed to the launders. Feed to the launders for this month was 135,000 tons.

Present Use of Cyclones at Vesta: By April 23, 1951, the value of the cyclone-vibrating screen dewatering system had become so apparent it was thought advisable to retain at least one of these circuits for emergency use. This was to be used in the event of a breakdown of one of the boot elevator units or to relieve some of the load from the boots if the overflow became too coarse for Dorr thickener feed. This circuit was put into operation in June 1951 when it became evident that the boat overflow was in reality too coarse for thickener feed and good filter operation. An installation consisting of eight 14-in. cyclones for thickening and classifying ⅛x0-in. launder middlings ahead of the Deister tables was also put into operation in June 1951. Since that time eight more cyclones have been added to this middlings circuit. These cyclones are operated with a larger than normal underflow opening to thicken the table feed slurry to about 33 pct solids, see Figs. 10, 11, and 12.

Ten 14-in. cyclones were installed to thicken and classify boot overflow in December 1951. The purpose of this installation is to divert coarse solids away from the Dorr thickeners and at the same time to improve filter operation. These cyclones are processing about 75 pct of the total boot overflow. The cyclones are mounted above vacuum filters that further dewater the underflow. Cyclone overflow goes to the Dorr thickeners, which in effect act as 3-in. cyclones in series with the 14-in. cyclones. Operation of this circuit has brought about marked improvement in filter operation and a reduction of solids in the plant recirculating water.

At present there are thirty-two 14-in. cyclones installed, of which 28 are in normal daily use, the remainder being spares. It is the opinion of the plant management that these cyclones are and have been of material value in maintaining capacity production and high operating efficiency in the fine coal plant.

Heated Screening

The Problem: It has been mentioned that the raw coal entering the plant is screened at ¼ in. over vibrating screens. The oversize or +¼-in. material goes to heavy media vessels and the −¼-in. is processed through Rheolaveur launders.

With these two methods of preparation a high screening efficiency is necessary to insure good cleaning performance. The heavy media vessels will not tolerate excessive amounts of fine coal in the separating medium and coarse, slabby material in the launder feed causes trouble by plugging the draw-off boxes.

For screening the raw coal at Vesta there are sixteen 6x14-ft double deck vibrating screens. The top deck is divided into three sections. The first section of the top deck is 1½-sq in. opening ¼-in. woven wire cloth, and the second and third sections are ¾-in. opening, 0.177 woven wire. The bottom deck has ¼x⅝-in. opening of 0.080 diam 18-8 stainless steel woven wire. The angle of the screens is 25°, RPM 1000, amplitude ¼ in.

It became apparent early in the operation of the plant that an effective screening job would be difficult primarily because of the relatively high mois-

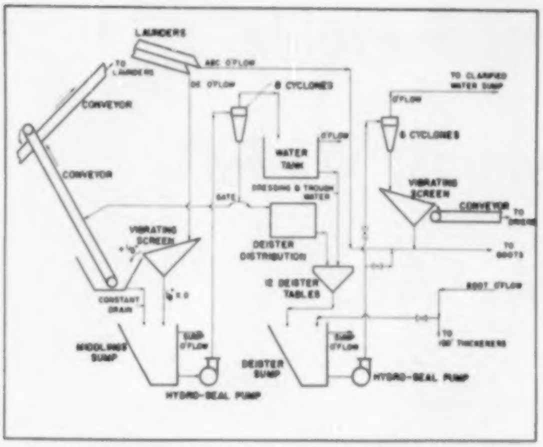


Fig. 11—Cyclones have been added since this data. There are now 32 installed.

ture content of the fines. This high moisture is due to water spraying as a safety measure to control dust during the mining operation. It was found that a clean ¼x⅝-in. screen opening would produce an effective ¼-in. separation on this coal. However, during an operating shift a gradual blinding of the screen cloth occurred. This necessitated manual cleaning of the screens by wooden mallet and brushing between operating shifts. At times blinding reached objectionable proportions during a shift. This required shutdown of certain units during plant operation with a resulting loss in production. This cleaning was costly not only in loss of production and increased labor but damage to screen cloth during this operation greatly reduced screen life.

Early attempts to avoid screen blinding included trial of several different types of screen cloth and single deck operation. A screen cloth with a long slotted opening proved effective in reducing blinding

Fig. 12—Bank of six cyclones operating in Deister clean coal circuit.

but permitted slabby material to enter the screen undersize. This interfered with launder operation. Single deck operation failed because the high percentage of refuse in the raw coal caused excessive wear on the fine wire cloth.

Finally in July 1953 heating equipment was installed on one of the 16 vibrating screens. Results of this experiment were so successful that at the present time all 16 of these screens are so equipped.

Heated Screening: Heated screening is an application of the fact that when an electric current meets resistance heat is created. Heat produced is proportional to the resistance of the conductor and to the square of the current. Thus if a low voltage, high amperage current is passed through screen wire cloth, a temperature rise in the cloth will follow.

The heating unit consists of a variable voltage transformer connected by flexible leads to the wire cloth. In practice these transformers range in size from 5 to 30 kva, with primary voltage of 220 and 440 and secondary voltage of 2 to 10 v. The transformers installed at Vesta are 16 to 36 kva, primary voltage 440 and secondary voltage of 4.5 to 9 v.

While heated screening is not the answer to all screening problems, it has been found very effective at Vesta, where blinding occurred because wet fines adhered to the screen cloth. The heated screen has the added advantage of being able to clean itself should a surge of wet mud pass over the screen.

When wet coal passes over a screen the initial build-up occurs by adherence of a layer of very fine wet particles to the screen surface. Particles are retained in this position by the surface tension of a thin film of water also present on the wire surface. Other fine particles adhere to these until finally the gap between the wires is bridged and blinding occurs.

The primary objective of heated screening is to eliminate the initial build-up. This is accomplished by inducing sufficient heat in the wire cloth so that any film of water adhering to the wire surface will evaporate rapidly. Any particles attempting to adhere to the clean dry wire are quickly detached by vibration of the screen.

Results of Heated Screening at Vesta: Power costs for the heated screening amount to about ¼¢ per ton of feed. This cost is more than offset by elimination of manpower used for manual cleaning and by increased screen cloth life. Records show that the unheated screen cloth averaged 560 hr of operation. This has been increased to about 2000 hr for the heated cloth.

Conclusion

Raw coal feed to the Vesta preparation plant averages 2100 tph, or more than 28,000 tpd of two 7-hr shifts. About 20,000 tons of clean coal are shipped to the steel plants each operating day. The plant produced about 4,145,000 tons of clean coal in 1953. Raw coal feed to the plant amounted to approximately 6 million tons for the same period.

Along with this high tonnage production, management is continually striving to produce better metallurgical coal at the lowest possible cost. The ultimate objective, of course, is to supply J & L blast furnaces the uniform high quality coke so necessary to iron production.

Discussion of this paper sent (2 copies) to AIME before Feb. 28, 1957, will be published in MINING ENGINEERING and in *AIME Transactions* Vol. 208.

Airborne Magnetometer Profile From Olympia, Wash., To Laramie, Wyo.

by W. B. Agocs and R. R. Hartman

IN the course of a return flight from Olympia, Wash., to Laramie, Wyo., an airborne magnetometer profile was recorded continuously. The level of flight was controlled at barometric levels along segments of the flight. Barometric altitude was changed from 3000 ft to a maximum of 12,000 ft above mean sea level to take care of topographic variations. Terrain clearance varied from 800 to 7300 ft. Horizontal positions were obtained by recording and tying to cities, towns, airports, river crossings, and other landmarks in the course of the flight.

Figs. 1-6 show the course of the flight and the general geologic and tectonic features traversed. The flight line begins south of Olympia, Wash., and proceeds south across the Puget-Willemette depression to the Columbia River. Thence it proceeds eastward along the Washington-Oregon border across the Cascade Mts. and lava flows and the Umatilla basin to Pendleton, Ore. From Pendleton it traverses the Blue Mts. uplift of Oregon in a southeasterly direction, and then to the Boise basin of southwestern Idaho. The flight line swings in an easterly direction as it passes over the Snake River downwarp to the south of the Idaho batholith. It then passes over the northern portion of the Great Basin and the heavily overthrust area at the intersection of the Utah, Idaho, and Wyoming borders. The flight then assumes an easterly course to cross the series of basins and uplifts of southern Wyoming and terminates near the western flank of the Laramie uplift.

It should be clearly understood that all statements in this article pertaining to the magnetics were interpreted on the assumption of elongate anomalies transverse to a single profile. Hence any theories or ideas advanced here are in line with future paths of research rather than trying to prove or disprove existing data.

Magnetic Interpretation: Depth determinations were made by the slope method of Peters,[1] which is stated as follows: "Draw a line in the maximum slope (at the point of inflection); draw a line whose slope is one-half this maximum slope; draw two lines parallel to this line tangent to the anomaly curve. The horizontal distance between the points of tangency is approximately equal to 1.6 times the depth [or the distance times 0.625]."

Certain assumptions must be made for determinations of the causes of the magnetic anomalies:[1]

1) The anomaly is caused by a susceptibility change rather than basement relief.

2) Polarization of the basement rocks is in the same direction as the earth's inducing field.

3) The disturbing magnetic mass has a great horizontal length in comparison to its width and its sides are vertical and of great vertical extent.

4) The anomaly is unique and not affected by the anomalies from nearby sources.

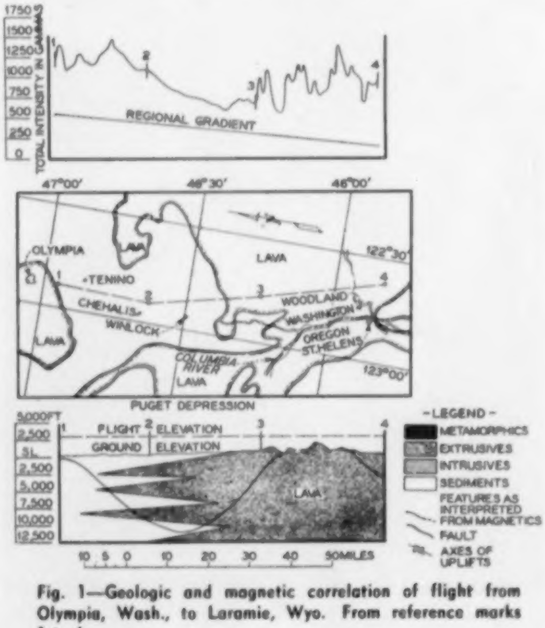
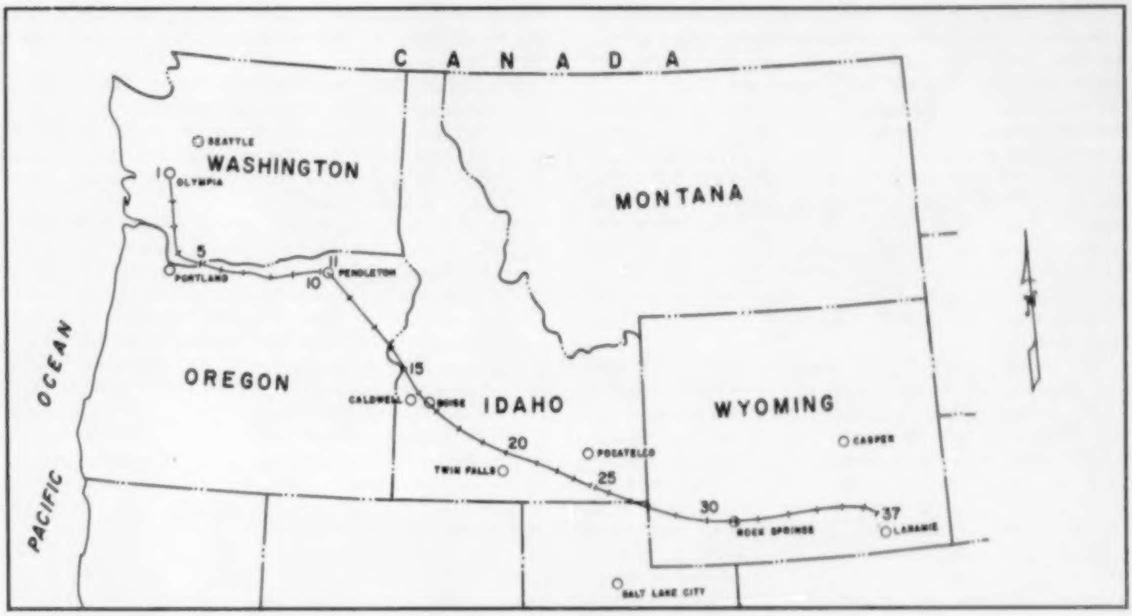


Fig. 1—Geologic and magnetic correlation of flight from Olympia, Wash., to Laramie, Wyo. From reference marks 1 to 4.

W. B. AGOCS is Chief Geophysicist and R. R. HARTMAN Geologist, Aero Service Corp., Philadelphia.

TP 4384L. Manuscript, Jan. 24, 1956. New York Meeting, February 1956.



These conditions are not always present in a single profile of this type.

Normal or Regional Gradient: The magnetic profile has not been corrected for the normal or regional gradient. The normal variation of the earth's total magnetic field is shown on Figs. 1-6, inclusive, as the regional gradient, as determined from Vestine et al.[3]

The intensity of the earth's magnetic field varies from 56,410 gamma at Olympia, Wash. to 56,980 gamma at Laramie, Wyo. The inclination is 70° north over the length of the profile.

A comparison of the observed regional gradient and the normal regional shows good general agreement, but there are some intense deviations that cannot be classified as being due to local anomalies. These will be discussed.

Geology and Magnetics, Olympia, Wash. to Pendleton, Ore. (Figs. 1 and 2): In the interval from reference marks 1 to 4, the magnetic profile is along the Puget depression. The rocks of this zone are predominantly Tertiary marine clastics and interbedded volcanics. The section is reported to be 20,000 to 36,000 ft thick. Maximum depth determination from the magnetics is found to be between 16,000 and 17,000 ft at reference mark 2. Other depths along this section range from 2000 ft at reference mark 1 to 3000 ft at reference mark 3 and at the surface between reference marks 3 and 4. It is extremely doubtful that these depths represent the basement.

Owing to interbedding of the volcanics in the area, numerous factors could be involved in reduction of these depths—such as the multiplicity of the overlapping sheetlike bodies of lava, or possibly a group of more basic flows somewhere near the horizon indicated by the depths. This will be referred to as the magnetic horizon rather than the basement, or some other specific cause.

The zone from reference marks 3 to 4 is characterized by sharp positive anomalies that probably are caused by the lava flows.

The magnetic profile from reference marks 4 to 10 swings to the east and crosses the Cascade Mts. and the Umatilla basin to the edge of the Blue Mts. uplift. The Cascade Mt. range is characterized by complex Miocene and Pliocene folding developed in metamorphosed Paleozoic and Mesozoic rock and batholithic intrusions of granodiorite. Tertiary volcanics covering the area are reported to be 5000 to 15,000 ft thick.

From reference marks 4 to 5 the magnetic anomalies are superimposed on the flanks of a massive positive divergence from the normal regional gradient. This divergence begins at reference mark 3 and will be discussed later. Isolated depth determinations show the magnetic horizon to be about 4500 ft midway between reference marks 4 and 5, and 2000 to 3500 ft immediately to the east of reference mark 5.

A major negative anomaly, on which are superimposed sharp positive anomalies, is recorded from reference marks 5 to 7 in the crossing of the Cascade Mts. The local superimposed positive anomalies of 230 to 650 gamma may be the result of basic intrusions, or possibly basic segregations from the parent acidic magma. To the west of reference mark 6 the depth to the magnetic horizon is 1900 ft and to the east it increases to 3000 ft.

Proceeding eastward, from reference marks 7 to 10, the magnetic anomalies are relatively sharp and of minor amplitude and width, indicating a surface cause. The level of these anomalies parallels the normal regional gradient. It is of interest to note that at reference mark 7 a sharp break in the magnetic level occurs. This is due to a susceptibility contrast of the order of 300×10^{-6} CGS units.

In summary, from reference marks 1 to 10, a maximum depth of 16,000 to 17,000 ft is recorded, but over most of the profile it ranges from 2000 to 4000 ft. This lack of agreement between the geologic and magnetic sections is almost certainly attributable to a magnetic horizon within the sedimentary section.

There are two more very interesting features of this profile—the previously mentioned broad divergences from the regional gradient, occurring between reference marks 3 to 7, inclusive.

From reference marks 3 to 5 a gamma positive regional deviation from the normal gradient is observed over a distance of 58 miles. From reference marks 5 to 7 a negative deviation of about 650 gamma is recorded over a distance of 36 miles. The former zone occurs at the margin of the Puget depression and the latter zone at the Cascade Mts. This type of anomaly is recorded at two other zones along the flight line. In view of the consistent indications the following is presented as one possible explanation.

The negative regional fluctuation over mountainous areas could be the result of deeper penetration of the granitic core of the mountain range into the sima layer of the earth's crust. This penetration could affect the sima by squeezing it into the margins of the mountain core. This would tend to thicken the basaltic layer along the edges of the trough where geosynclinal sediments are being deposited. The thickening of the basaltic layer, which has a higher susceptibility, would produce the regional high over the flanks of basins and the complementary thinning would produce a regional low over the mountain system.

A second factor, however, that must be considered is that the Curie point geotherm would be 12 to 15 miles deep. At this level the magnetic properties of magnetite would be lost. The four such anomalies described in this article are fairly consistent with this depth if calculations are based on ¼ of the breadth of the anomaly. The errors would arise, of course, in the method of determining the depth and the elongation of the anomalies due to the flight direction.

Pendleton, Ore., to Mountain Home, Idaho (Fig. 4): The profile from reference marks 11 to 18 extends southeastward from the edge of the Umatilla basin, across the Blue Mts. uplift, and then skirts southwest of the edge of the Idaho batholith along the Snake River downwarp.

The Blue Mts. uplift is a group of dissected lava plateaus. The volcanics of the area are partly of Carboniferous age and are highly metamorphosed. The Carboniferous rocks, known as the Burnt River Schists, are 5000 ft thick or more. Intrusives in the form of granitic blocks are exposed over much of the area and are associated with the Idaho batholith and others extending into British Columbia. Intrusions of peridotite, gabbro, and diorite are associated with the granitic masses. A system of normal faults trends approximately east-west in this area (parallel to the flight path). The area is also mineralized to some extent, gold, scheelite, and molybdenum being associated with the intrusions.

The magnetic anomalies in this section from reference marks 11 to 18 are markedly different from those shown on the previous section. The general magnetic gradient is roughly parallel to the normal regional gradient. However, from reference marks 11 to 13 a major positive high of 300 gamma, on which local anomalies of up to 500 gamma are recorded, extends over a distance of 60 miles. This zone, which is located in the approach to the Blue Mts. uplift from the Umatilla basin, is not followed by a negative zone as in the passage from the Puget depression to the Cascade Mts.

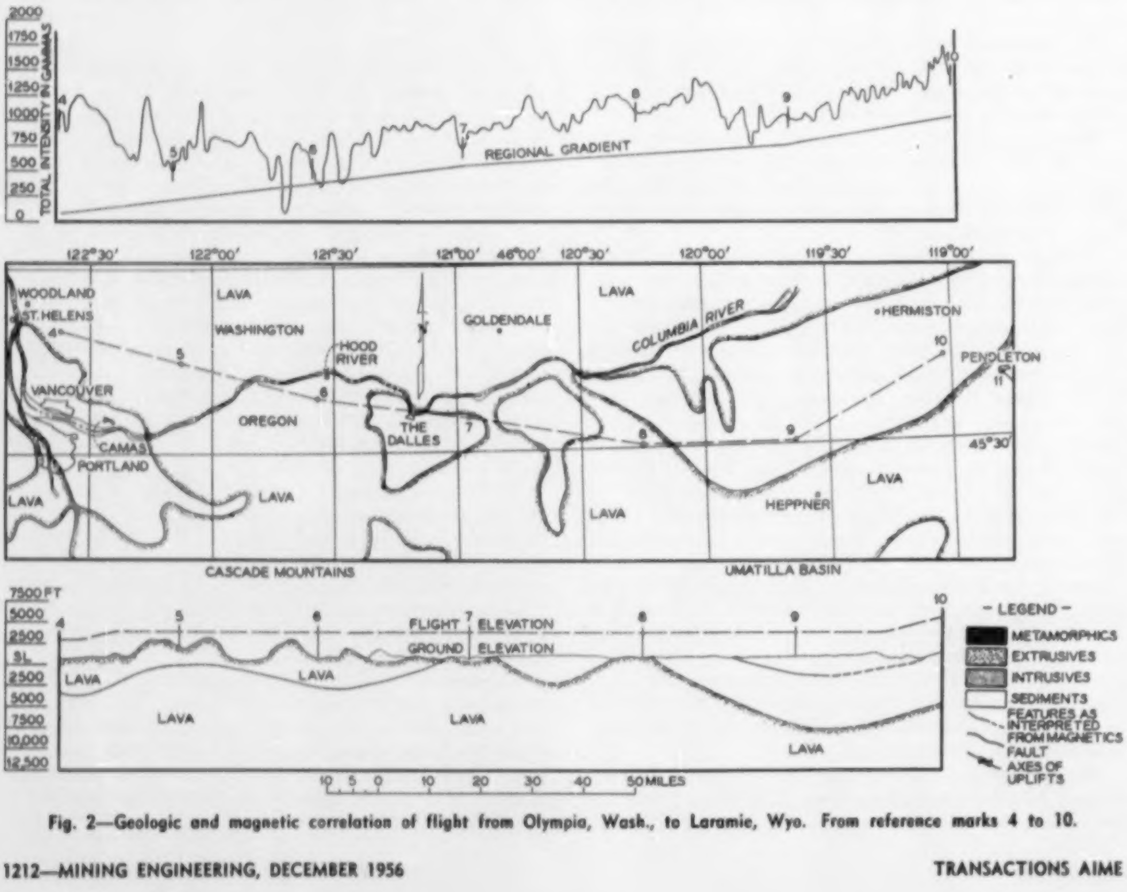


Fig. 2—Geologic and magnetic correlation of flight from Olympia, Wash., to Laramie, Wyo. From reference marks 4 to 10.

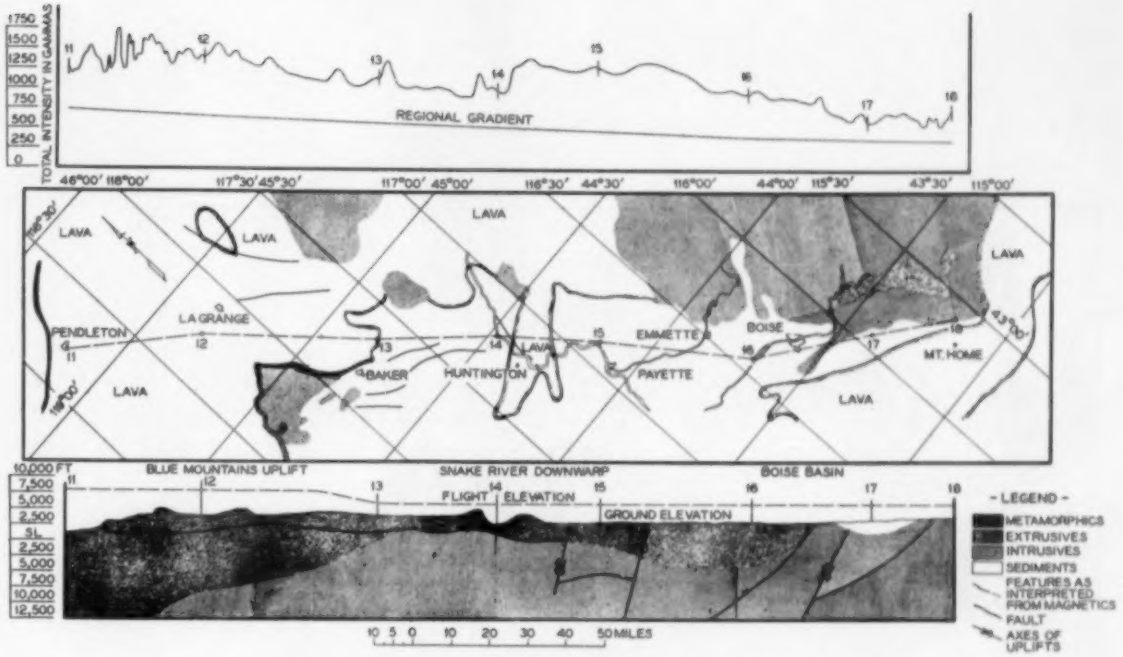


Fig. 3—Geologic and magnetic correlation of flight from Olympia, Wash., to Laramie, Wyo. From reference marks 11 to 18.

The minor anomalies in the zone from reference marks 11 to 12 show a depth of the magnetic horizon from the surface to 3000 ft subsurface. In the section from reference marks 12 to 14 the magnetic horizon lies at sea level, or 4000 to 5000 ft subsurface. At reference mark 14 a second *break* is observed where a 300 gamma positive anomaly discontinuously marks the approach to the Snake River downwarp and the flanks of the Idaho batholith. The Snake River downwarp lies south of the batholith and is covered by Tertiary lavas. The batholith forms the basement and is composed of quartz monzonite, although marginal differentiates are common. Recently it has been suggested that the batholith was originally a diorite but was altered by ascending siliceous solutions. Dikes intrude in several places.

Mineralization in the form of cinnabar is found north of reference mark 17. Gold, silver, and some bismuth are mined near Boise. The ores are associated with the dikes. Iron of economic value has never been found, but fragments of magnetite occur in gravels northwest of Boise.

The anomalies from reference marks 14 to 18 are markedly different from those shown on the previous section. They are smooth and of 100 to 300 gamma in amplitude. East of reference mark 14 the depth to the magnetic horizon increases to 10,000 ft to 13,000 ft subsurface, and it increases to 18,000 ft subsurface about 10 to 12 miles southeast of reference mark 15. This indicates two sectional breaks in the vicinity of the Snake River downwarp that could be due to faults of 9000 and 5000 ft respectively. Proceeding southeastward, to the southeast of Boise, the depth decreases to 2300 ft. It then abruptly increases to 11,000 ft in the vicinity of reference mark 17 (due to faulting), decreases to 1700 ft near reference mark 18, and rises to the surface at reference mark 18. The major fault at point 17 is probably on the flank of the Idaho batholith.

Mountain Home, Idaho, to the Green River, Wyo. (Figs. 5 and 6): The section from reference marks 18 to 30 is shown in Figs. 5 and 6. This portion of the profile continues across the Snake River downwarp, thence across the Great Basin or Basin and Range province, the Bannock, Crawford, and Absaroka overthrusts, and across the Green River Basin to the flank of the Rock Springs anticline.

From reference marks 18 to 21 the flight continues across the Snake River downwarp. The magnetic profile along this segment of the flight shows a zone of low amplitude, high frequency anomalies. The depth to the magnetic horizon rises from 4900 ft at reference mark 18 to 1600 ft to the east of reference mark 19. A fault of 4000 to 5000 ft is indicated between reference marks 19 and 20 at Wendell. At reference mark 20 a sharp 150 gamma discontinuity is recorded. This level change occurs in the approach to the Great Basin.

In the interval from reference mark 20 to 22 the magnetic anomalies are broad and of 150 to 200 gamma in amplitude, on which are superimposed the minor 10 to 50 gamma anomalies of relatively high frequency. The magnetic horizon rises to a level of 3700 ft midway between reference marks 20 and 21 and then rises to the surface immediately to the west of reference mark 22. In the vicinity of reference mark 22, the depth increases to almost 13,000 ft. This is a major horst zone between reference marks 21 and 22, a basement high flanked by faulting.

The Great Basin province is a series of north-south trending ranges of sedimentary rock overlain by patches of lava. Sedimentary strata up to 40,000 ft have been estimated in the deepest part of the basin near the flight line.

From reference mark 22 the depth to the magnetic horizon increases from 4500 ft to over 35,000 ft between reference marks 23 and 24. This latter level rises to the surface east of reference mark 26 near the town of Preston, Idaho. From reference marks 22 to 27 the recorded gradient follows the normal

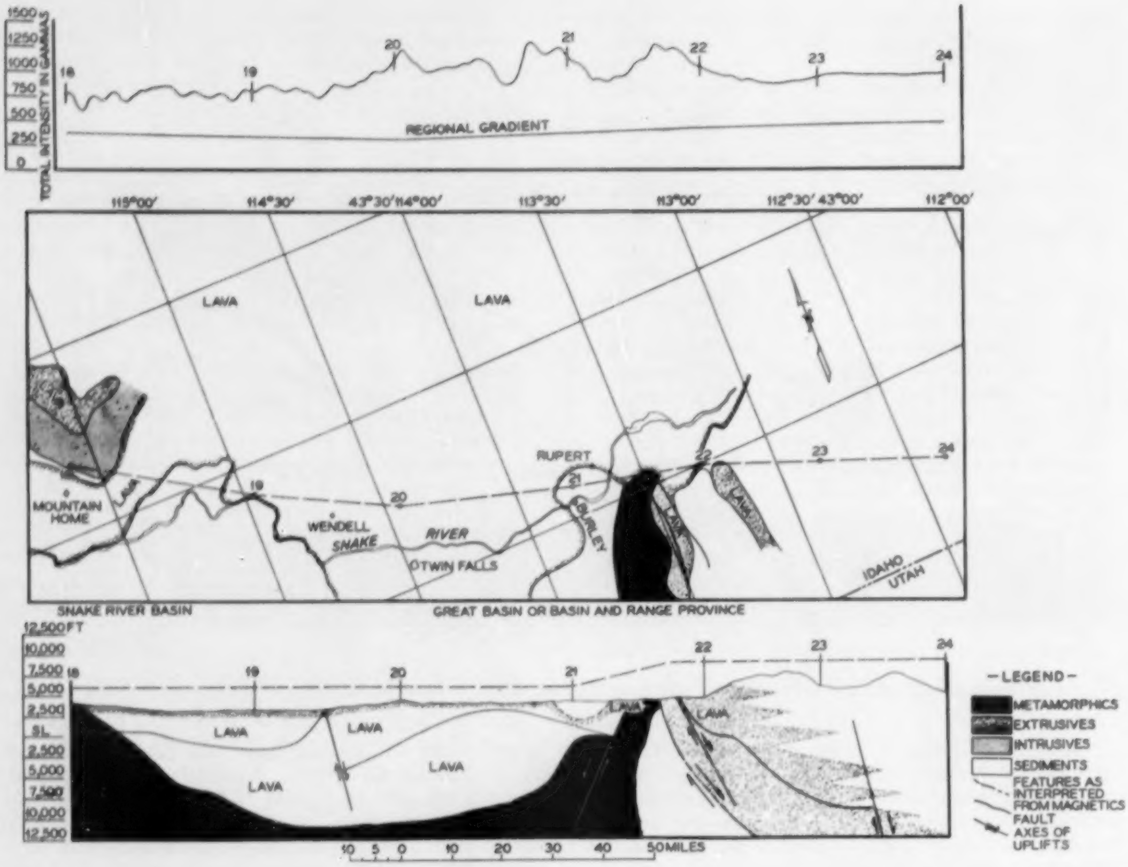


Fig. 4—Geologic and magnetic correlation of flight from Olympia, Wash., to Laramie, Wyo. From reference marks 18 to 24.

regional gradient and the anomalies are of low amplitude.

From reference marks 27 to 29 the flight crosses the overthrust zone of the Bannock, Crawford, and Absaroka faults. The area had been folded prior to faulting and the large folds are truncated by the nearly horizontal thrust sheets. The rock is composed of thick geosynclinal sedimentary deposits. This area, known as the Green River basin, is continuous to reference mark 30, where the flight approaches the western flanks of the Rock Springs anticline. The Green River basin is believed to contain about 30,000 ft of sediment.

To the east of Bear Lake, from reference marks 27 to 30, depths to the magnetic horizon range from 25,000 to 29,000 ft—in good agreement with the estimated geologic sectional thickness. From reference marks 27 to 30 a positive deviation from the normal regional is recorded. This deviation lies over the overthrust zone of the Green River basin.

Green River to Laramie, Wyo. (Fig. 7): This section (shown in Fig. 7) from reference marks 27 to 37 extends from the Green River to its termination at Laramie. This section of the profile crosses the Rock Springs anticline, Washakie basin, Rawlins uplift, Hanna basin, and Laramie basin.

Between reference marks 30 and 32 the Rock Springs anticline is cut by a series of faults perpendicular to the axial trend. The Leucite Hills forming the north end of the anticline are remnants of

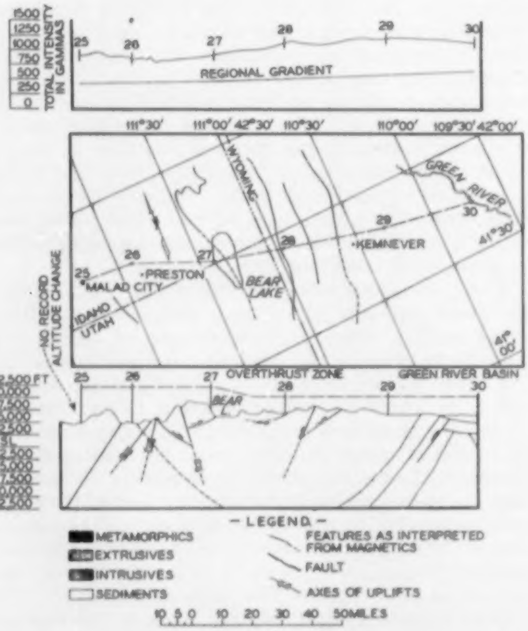


Fig. 5—Geologic and magnetic correlation of flight from Olympia, Wash., to Laramie, Wyo. From reference marks 25 to 30.

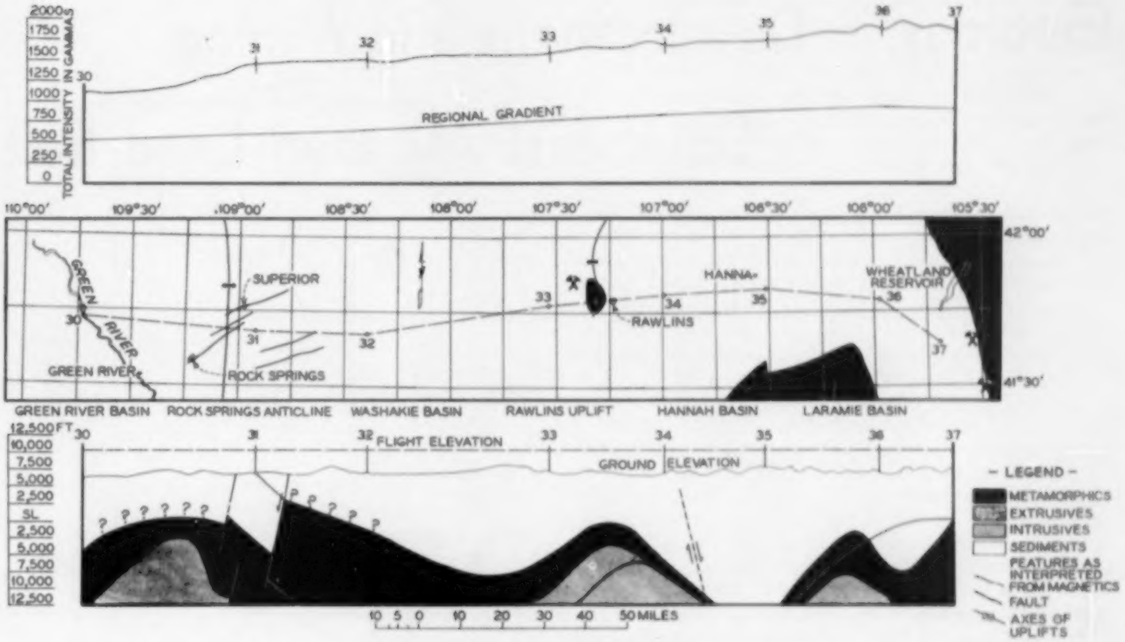


Fig. 6—Geologic and magnetic correlation of flight from Olympia, Wash., to Laramie, Wyo. From reference marks 30 to 37.

cinder cones and lava sheets. The recorded magnetic gradient is the same as the regional gradient, except for the 300 gamma discontinuity immediately to the west of mark 31. The discontinuity occurs in the faulted zone between the axis of the Rock Springs anticline and the Washakie basin. This discontinuity marks the location of a fault, since the depth to the west is over 38,000 ft and in the vicinity of reference mark 31 the depth is 17,000 ft. This would mean a fault of over 20,000 ft. This great depth lies on the flank of the Rock Springs anticline.

The flight passes from this fault zone to the Washakie basin. The thickest and deepest part of the Washakie basin, near its eastern flank, is estimated to be 30,000 ft. At reference mark 32 the depth to the magnetic horizon ranges from 24,000 ft to 17,600 ft, and at reference mark 33 the depth is found to be 20,000 ft. This is the Washakie basin up to the Rawlins uplift.

The Rawlins uplift is characterized by high structural relief complicated by folds and thrust faults. One thrust fault has pushed Pre-Cambrian metamorphics over Eocene sediments which outcrop east of the town of Rawlins. The Hanna basin lies east of the Rawlins uplift and although small, it contains a great thickness of sediments. The magnetic control bears out the basement control of the Rawlins uplift since, in the interval from the town of Rawlins, between reference marks 33 and 34 to reference mark 34, the depth to the magnetic horizon ranges from 12,000 to 14,000 ft. This shows an uplift of 8000 to 9000 ft above the general floor of the Washakie basin. The magnetic horizon drops to 26,000 ft between reference marks 35 and 36 in the Hanna basin. This latter depth is somewhat less than the estimated sedimentary thickness of the Hanna basin of 35,000 ft.

The Laramie basin is a syncline between the Laramie range on the east and the Medicine Bow range on the west, both of which are formed from Pre-Cambrian crystallines. The basin contains about 8000 ft of Carboniferous and Mesozoic sediments. There is some mineralization in both the Laramie and the Medicine Bow ranges. The depths, as computed from the magnetic record, rise continuously from the Hanna basin to the end of the profile, where the depth to the igneous basement is found to be 8500 ft subsurface. No indication is found of an uplift separating the Hanna basin from the Laramie basin as the result of the northward continuation of the Medicine Bow range or the southward continuation of the Sweetwater uplift.

Conclusions

The 1000-mile magnetic profile yielded numerous general magnetic correlations with broad geologic structures—in particular, the positive deviations on the flanks of basin areas and the negative deviations over mountain areas—and of local structures and depth to igneous rocks. This information could be used as a basis for expanded surveys in basin areas for petroleum, or structural areas for minerals, and numerous academic studies.

Acknowledgments

The authors wish to express their gratitude to J. J. Ardiff and W. Lucas of the Magnetometer Div. of Aero Service Corp. for the preparation of the illustrations used in this article. Permission to publish this article was granted by C. H. Frost and by Virgil Kauffman, President of Aero Service Corp.

References

[1] L. J. Peters: The Direct Approach to Magnetic Interpretation and Its Practical Application. *Geophysics*, 1949, vol. 14, No. 3.

[2] E. H. Vestine, L. Laporte, I. Lange, C. Cooper, and W. C. Hendrix: Description of the Earth's Main Magnetic Field and Its Secular Change, 1947. Department of Terrestrial Magnetism, Carnegie Institute of Washington.

[3] A. J. Eardley: *Structural Geology of North America*, Harper & Bros., New York, 1951.

[4] USGS Geologic Map of the U. S., 1932. Scale 1:2,500,000.

[5] AAPG Tectonic Map of the U. S., 1944. Scale: 1:2,500,000.

Discussion of this paper sent (2 copies) to AIME before Feb. 28, 1957, will be published in MINING ENGINEERING and in AIME *Transactions*, Vol. 208.

Geology in Development and Mining, Southeast Missouri Lead Belt

Recommended development for three classes of orebodies

by Frank G. Snyder and John A. Emery

MINING geology has a threefold objective: to guide prospecting for new ore, to evaluate known orebodies as development risks, and to supply the detailed knowledge of ore structures necessary for more economical extraction. The role of geology in exploration is well established. This article will deal solely with orebody evaluation and the application of geology to development and mining problems as practiced by St. Joseph Lead Co. in the southeast Missouri district, which for many years has supplied approximately 30 pct of U. S. lead production. The primary concern of the present discussion is geology in the district, not geology of the district.

The southeast Missouri district, locally called the Lead Belt, includes parts of St. Francois, Madison, and Washington counties. The first recorded mining was about 1720 at Mine La Motte in Madison County on surface showings of lead ore. Later discoveries near Bonne Terre, twenty-five miles to the north, led to extensive prospecting and underground mining. To date the district has yielded well over 8 million tons of pig lead, valued at more than $1 billion. By far the greater part of the lead production has come from an area of 50 sq miles, centering on the city of Flat River, as shown in Fig. 1.

Role of Geology in Metal Mining

The history of the use of geology in the older metal mining districts follows a simple consistent pattern that may be considered in terms of four distinct phases. Each geologic phase is a response to the economic status of the mining district; each has definite characteristics setting it apart from other phases; and each passes into the succeeding phase as the mining operation gradually exhausts the large, rich, easily discovered deposits and lives upon increasingly smaller, lower grade orebodies that are less easily found.

The geologic phases and their major characteristics are as follows:

F. G. SNYDER and J. A. EMERY are Geologists, St. Joseph Lead Co., Bonne Terre, Mo.

TP 43911. Manuscript, May 4, 1956. New York Meeting, February 1956.

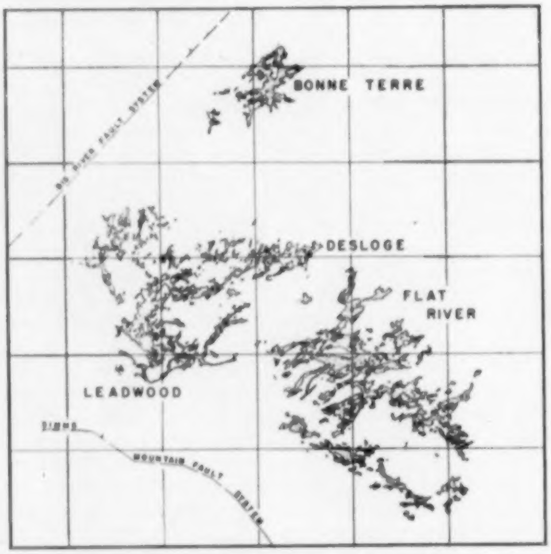
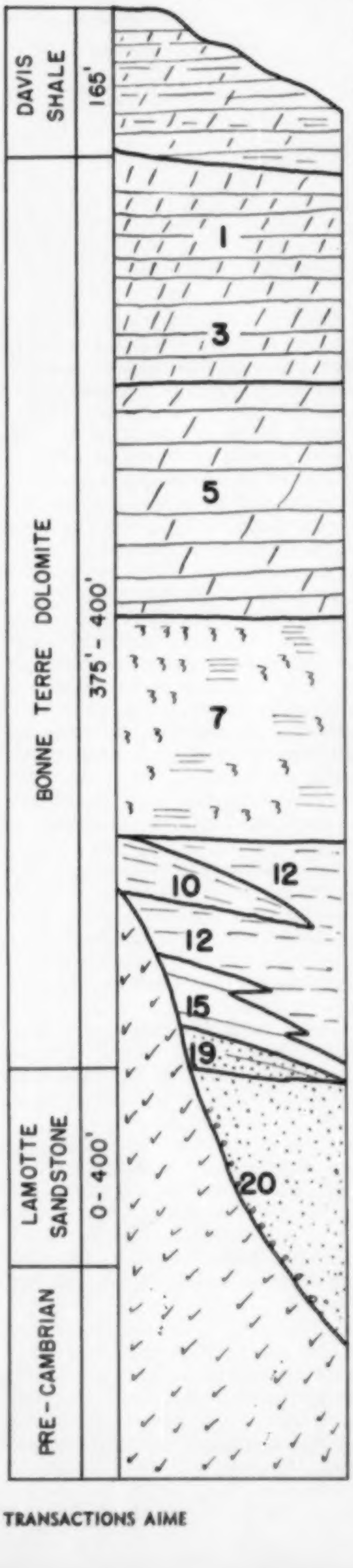


Fig. 1—Central portion of southeast Missouri lead district.

Period of No Organized Geologic Program: During the early history of a great metal district ore is easily found by random prospecting and wildcat drilling. Many small operators are active, as profitable mining can be done with a relatively small capital outlay and a minimum of technical staff. Mining practice consists largely of sinking a shaft on an ore showing and following the ore in sight. When the ore is exhausted or becomes unprofitable to mine, the process is repeated on a new ore showing. The miners become familiar with certain ore habits and use this practical geologic knowledge to guide mining.

The method is successful as long as large tracts of virgin mineralized area are available. This phase ends when small-scale mining methods are no longer profitable and the mines become consolidated into the hands of a few operators.

Fig. 2—Generalized stratigraphic section.



1—Dense, gray dolomite.

3—Mottled blue and white dolomite. Chocolate brown dolomite at Bonne Terre.

5—Thick-bedded tan dolomite, often oolitic.

7—Main algal reef zone. Commonly thick massive reef throughout on structural center in Federal area. Thin and scattered reefs in Leadwood area. Non-reef rock is gray-spotted, tan or crepey.

10—Thick, fine-grained, even-bedded tan on southwest side of district. Coarse-grained, lenticular tans over structural centers. Thin or absent between structural centers.

12—Thin-bedded, fine-grained, crepey and spotted dolomite. Contains frequent tan beds. Marble beds common at base. Thin over structural centers.

15—Light colored, coarsely crystalline. Often glauconitic. Frequently thick over structural center.

19—Tan or gray sandy dolomite. Usually thicker at structural centers.

20—Sandstone with occasional siltstone beds. Poorly cemented in upper part. Thin dolomitic sand beds in upper part. Residual igneous boulders at base. Complex of granites and porphyrys.

In the southeast Missouri lead district this first phase was a long one, continuing into the early 1920's. An important development of this period was extensive use of the diamond drill in prospecting. Adequate ore reserves could be maintained by low cost diamond drilling; geological assistance was considered unnecessary. The only important geological contribution of this period was the Buckley report of 1908.[1]

Period of Limited Geologic Program: This phase is characterized by an organized exploration program under a trained geologist or an engineer familiar with geologic problems. Discovery of new ore is the main objective. Geologic information is catalogued, organized, and used to guide further exploration. Delineation of geologic theory in the district is left to part-time consultants, government survey agencies, and occasional visitors.

Mine operators and geologists alike seldom recognize that geology may make a material contribution to economic mining; hence little attempt is made to apply geology to operating problems. Application of geology to mining during this phase is largely rendered by engineers and miners whose growing knowledge of ore habits is an important asset to successful mining.

The second phase is marked also by increased consolidation of individual ventures and the emergence of a few units or a single big operation which can finance the costly equipment and large-scale methods needed to handle larger tonnages of lower grade ore. The greatest production of metal per unit of time usually comes during this phase.

This phase draws to a close as ore becomes harder to find, as the mine operation must turn to lower grade ore and smaller orebodies, and as a forward-looking management recognizes that detailed geologic study is necessary to maintain an adequate mining reserve.

The second phase of Lead Belt geology was characterized by expanded use of the diamond drill in prospecting and the development of the St. Joe shovel as the basic piece of mining equipment. In few mining districts have operations been geared so closely to particular methods as were the exploration and ore loading in southeast Missouri. In this period R. E. Wagner and J. E. Jewell, who guided the exploration program, subdivided the ore-bearing Bonneterre formation into zones. Their classification, with minor modifications, continues to be the basis of the core-logging system and the underground mapping program.

Period of Detailed Geologic Study: The change from the second phase to the third is usually the only one in the sequence for which a definite date can be picked. This is the date a department concerned with mining geology is established.

The third phase is marked by a systematic mapping program and an intensive search for factors controlling localization of ore. This is a period of research. Mined out areas are mapped and studied and hypotheses developed to relate and explain certain features. At this time the geologist learns which features in the rocks are significant in the interpretation of ore occurrence and which, regardless of theoretical interest, bear no relation to ore.

The southeast Missouri district embarked on this phase with the organization of a separate geological department by the St. Joseph Lead Co. in 1946. This department was directed by John S. Brown, who instituted an intensive program of mapping

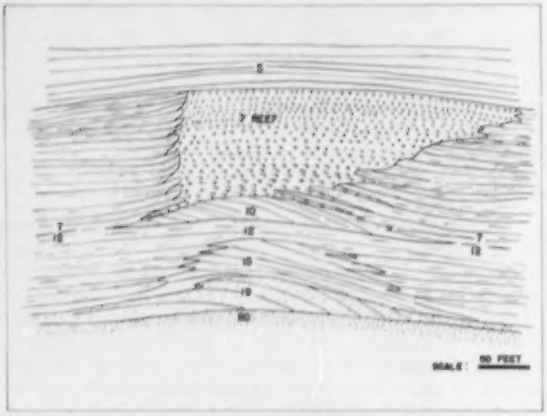


Fig. 3—Cross section of a structural center.

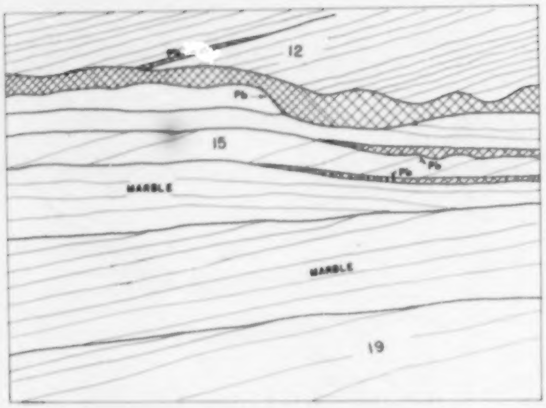


Fig. 4—Bedded ore at 12/15 contact.

mine workings, a study of geologic controls of orebodies, and the application of geologic knowledge to prospecting, development, and mining problems.

Period of Application of Geology to Daily Mining Operations: In practice, the third phase passes gradually and slowly into the fourth, since geologic knowledge is put into use almost as soon as acquired. Mapping and study of mined areas is a continuing process. As new knowledge is gained and interpretations are improved, areas studied earlier must be restudied and re-evaluated in light of the advanced knowledge.

The fourth phase begins when the mine development program consistently fails to sustain the predetermined level of production. In the mining operation this period is characterized by increasingly greater risks in developing smaller and lower-grade deposits and by searching out and mining crumbs of ore passed over when large rich orebodies were readily available. It is a period when every engineering and scientific tool must be used to maintain the competitive position established during the early history of the district. Prolonging this period is a major objective of geology as applied to mining.

Geology of the Lead Belt

Geologic Setting: Recognition of the major geological features of the southeast Missouri lead deposits is preliminary to further consideration of orebodies. More detailed accounts of geology of the lead deposits have been reported by Ohle and Brown[2] and Tarr.[3]

The lowest Upper Cambrian sediments represent a normal orthoquartzite-limestone sequence deposited on Pre-Cambrian porphyry and granite. The old basement surface presented an irregular, rolling topography, with local relief of several hundreds of feet. On this irregular floor, filling the valleys and feathering out on the flanks of igneous knobs, rests the Lamotte sandstone. Overlying the Lamotte with minor local disconformity is the ore-bearing Bonneterre formation.

The Bonneterre formation within the Lead Belt area is believed to have been deposited in a shallow water environment, probably much like that of the shallow marine banks of the Bahamas region of the present day. Although probably deposited initially as limestone, large parts of the formation, especially near the orebodies, are now dolomite.[3]

Over the Bonneterre *banks* minor differences in water depth existed as a result of undulations on the Lamotte surface. Igneous knobs, protruding as islands in the Upper Cambrian sea, channeled and guided currents, further emphasizing the bottom irregularities. Minor depth differences on the shallow irregular sea bottom of lower Bonneterre time provided a variety of sedimentary environments resulting in extensive local facies differences and a wide range of sedimentary structural features. The major rock types are tan crystalline—often oolitic—dolomites, representing well sorted carbonate sands; the gray, shaly and mottled dolomites, representing unsorted carbonate sand and mud with clay minerals; and organic deposits, representing reefs in place and recognizable organic detritus. Irregular and intermittent subsidence of the area as the seas encroached on the land mass to the southwest gave rise to vertical differences resulting in alternation of tan and gray rocks in the Bonneterre sequence.

The Bonneterre formation has been subdivided into units, designated 1 through 19, as shown in Fig. 2.

The major cyclical repetition, prominent in the lower Bonneterre, includes the tan crystalline 19, 15, 10, and 5 zones interbedded with the gray, shaly 12 zone and gray nonorganic parts of the 7 zone. Occasional thin gray dolomite beds in the tan units and tan beds in the gray units record minor local differences in sedimentation.

A widespread development of organic structures occurs at and near the base of the 12 and 15 zones and throughout the 7 zone. Those in the lower part of the section, locally called *marble*, include small bioherms and thin beds of organic detritus.

The 7 zone algal reefs grew on favorable sites over the carbonate sand banks. Individual reefs, usually striking northeast in the central productive area, range in size up to 1000 ft in width and up to three miles in length. Although fairly consistent in growth pattern within a small area, considerable differences in minor features of reefs exist in different parts of the district.

Major structural features of importance to the central productive area are the Farmington anticline on the northeast side of the district and the Simm's Mountain fault on the southwest side of the district.[3,4] The effect of this combination of structural elements is a regional dip of nearly 2° to the southwest, which places the base of the orebearing

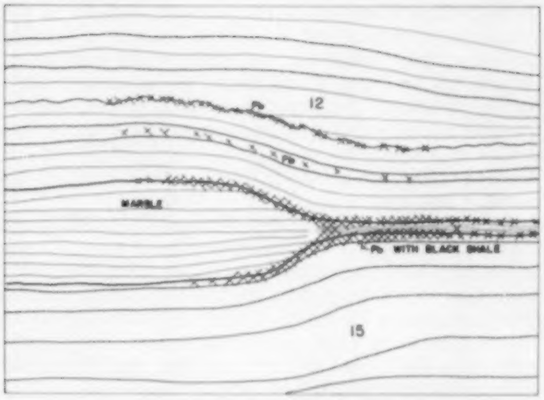


Fig. 5—Bedded ore with *marble fan.*

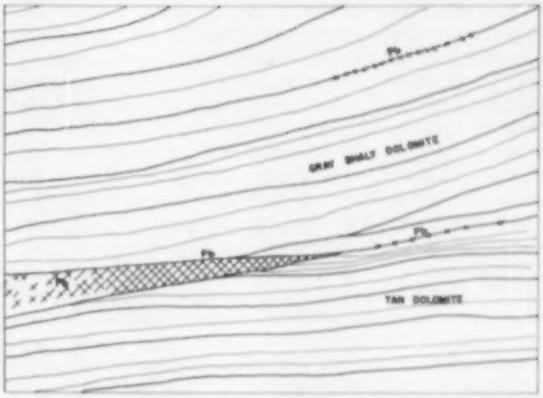


Fig. 6—Bedded ore on flank of structural center.

formation approximately 400 ft lower in the southwesterly mine workings than it is to the northeast.

Within the mined area several faults of 40 to 100-ft displacement cut the orebodies and have an important bearing on mining. Many lesser faults of a few feet displacement are related to the major faulting.

Geology of Orebodies: Throughout the lower half of Bonneterre time certain areas became topographic highs on the sea bottom which often persisted. The highs, generally trending northeast, exhibit arch structures at one or more horizons owing to a greater abundance of tan carbonate sand which grades into and intertongues with gray, fine grained sediment on the flanks. These topographic highs were the sites in which many sharp contacts between different kinds of sediment were formed and were the banks on which algal reef growth was most abundant. They are the areas that contain many sedimentary structures favorable for ore deposition. Hence they are termed *structural centers.*

Probably 90 pct or more of the ore mined within the district has come from within and along the flanks of the structural centers. A diagrammatic cross section of a well developed structural center is shown in Fig. 3. In some parts of the district a strong structural center may show almost continuous mineralization for more than 200 ft vertically, through all zones from the 19 up into the 5 bed. Elsewhere in the district only certain horizons are mineralized, or the structure itself may have limited height and ore may be confined to a single stratigraphic zone.

Bedded orebodies are those in which the ore is concentrated along bedding planes or disseminated as replacement of bedded dolomite within a restricted vertical zone. Ore frequently is associated with thin beds of black shale. These orebodies occur in a variety of sedimentary structures resulting from interruptions or discontinuities or changes that took place during deposition of the Bonneterre formation. Several types of bedded orebodies are shown in Figs. 4 through 7. Attitude of the floor of the orebody, a factor of paramount importance to mining, depends on the type of surface on which the ore-bearing bed was deposited. The floor may be a flat-lying bed, an arch crest or flank, a porphyry slope, or any minor sea bottom irregularity.

Reef orebodies are those within or marginal to recognizable organic structures of the 7 zone. A reef is defined as a major organic mass, in place, composed of algal deposits and entrapped and interbedded sediment. Where best developed it is roughly comb-shaped in plan, the teeth extending to the southeast as shown in Fig. 8. The major ore-controlling structure within the reef, termed a *roll,* is a rectilinear unit composed of a succession of superimposed colonial growths separated from similar units by clastic sediment or growth lines. Rolls vary from a few feet to a few tens of feet in width and

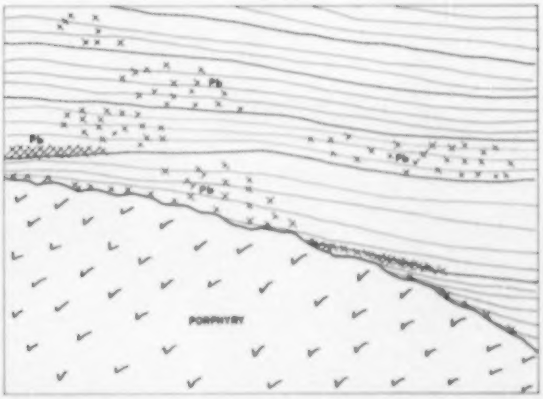


Fig. 7—Bedded and disseminated ore at dolomite/porphyry contact.

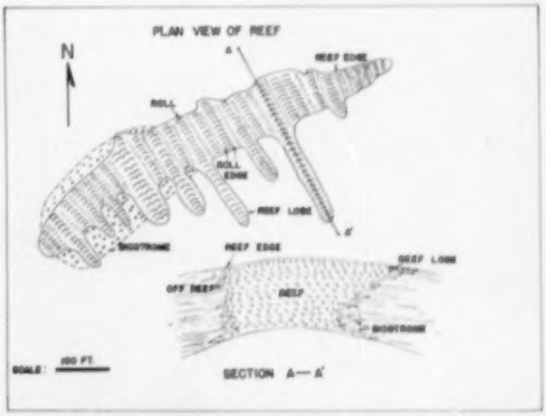


Fig. 8—Plan and section of 7 bed reef.

may extend the full width of the reef. Other features recognized as specific types of ore structures include the reef-edge, the precipitous contact of reef and clastic sediment with extensive intertonguing of reef and nonreef rock; the off-reef, the bedded deposit adjacent to the reef mass and dipping away from the reef; the reef lobe, a long extension of a roll or a series of rolls away from the reef; and the biostrome, a bed of organic material unbroken by growth lines or clastic sediment. Figs. 9 and 10 show examples of reef orebodies.

Reef orebodies present many mining problems not found in bedded deposits. Even in strongly mineralized areas large parts of the reef are barren. Ore is commonly more abundant along roll edges than within rolls. Adjacent pay holes may be on the same orebody or may be separated by considerable waste rock. Mining practice must be based on width of roll and width of inter-roll zone. Complete extraction of ore without excessive dilution requires careful attention to roll habits.

Evaluation of Orebodies

As orebodies may occur anywhere within a stratigraphic interval of nearly 300 ft and in widely different structural settings, each orebody is evaluated as a development risk by the mine geologist. Information gained from underground mapping and from study of drill logs provides the basis for evaluation.

Drillhole Data: The core drilling system followed in the district[4] includes detailed logging of rock types in drill core and discarding all core except that which is mineralized. Mineralized core is split; half is assayed and half saved for future reference.

In evaluation of orebodies, core logs of holes in the orebody and adjacent holes are studied for details of lithology of the host rock and of beds above and below the ore. Character of the rock which will form the mine back is noted. Consideration of logs of holes adjacent to the orebody is necessary to define facies changes, initial dip of beds, and probable width of orebody. The mineralized portion of the drill core is examined for character and distribution of ore, character of the host rock, extent of post-ore leaching, and extent of grinding core.

Position of the Orebody on Structure: A major factor in evaluation is the position of the orebody relative to the structural center. Very few unproductive developments are cut within a structural center. Unproductive developments, studied in the course of mapping old workings, usually represent scattered pay holes on the fringes or outside of structural centers. Care must be exercised with those orebodies that are on structure but beyond the limits of extensive mineralization; there are many examples of good structure and favorable looking rock which bear weak mineralization or none at all.

Within a structural center some contacts and some zones are generally better mineralized than others. However, beds sometimes regarded as unfavorable may carry good ore when well located on structure. The occurrence of many stopes of 100 to 200 ft of mined height through all stratigraphic zones indicates that rock type alone cannot be regarded as an unfavorable factor in evaluating an orebody.

Production History of Ore Horizon in the Area: Certain ore horizons may be productive in one part of the district and unproductive in another. Production history of the stratigraphic horizon in the immediate area of the orebody being evaluated is considered. Stratigraphic zones with poor production records are considered unfavorable, although they may show occasional good pay holes.

Rating the Orebody: Orebodies are rated according to decreasing mining probability on a scale of 1 through 5, as shown in Table I. The scale is a measure of the risk entailed in developing an orebody. No. 1 on the scale is essentially a sure thing; No. 5 is equally sure to be unproductive.

Most of the items on the scale involve little subjective judgment. The requirement of return relative to development cost embraces those intangibles gained from study of drill core and first hand knowledge of the area and is carefully weighed for each orebody.

Recommended Development

Orebodies Classified According to Mining Habit: Classification of orebodies by geological features, although necessary in interpretation, is of little direct value to operators. It is more useful to classify according to mining habit. By mining habit is meant those characteristics of an orebody, particularly size, shape, and nature of the mining floor, that determine the development and mining procedure necessary for most complete and economical extraction of the ore. Variations in mining habit of Lead Belt orebodies are due to minor, but usually predictable,

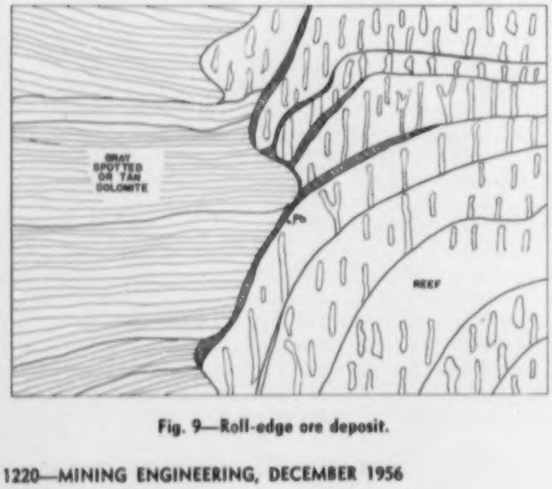


Fig. 9—Roll-edge ore deposit.

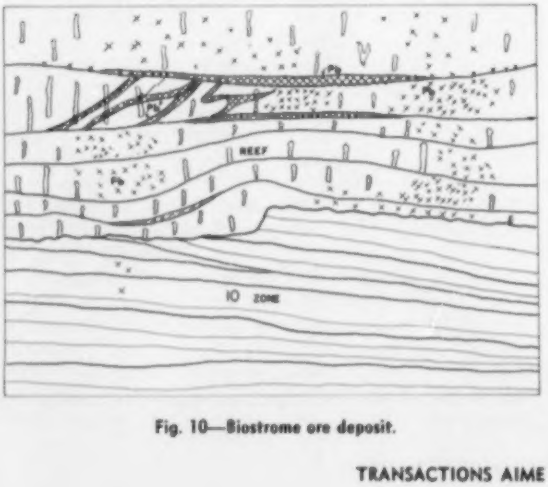


Fig. 10—Biostrome ore deposit.

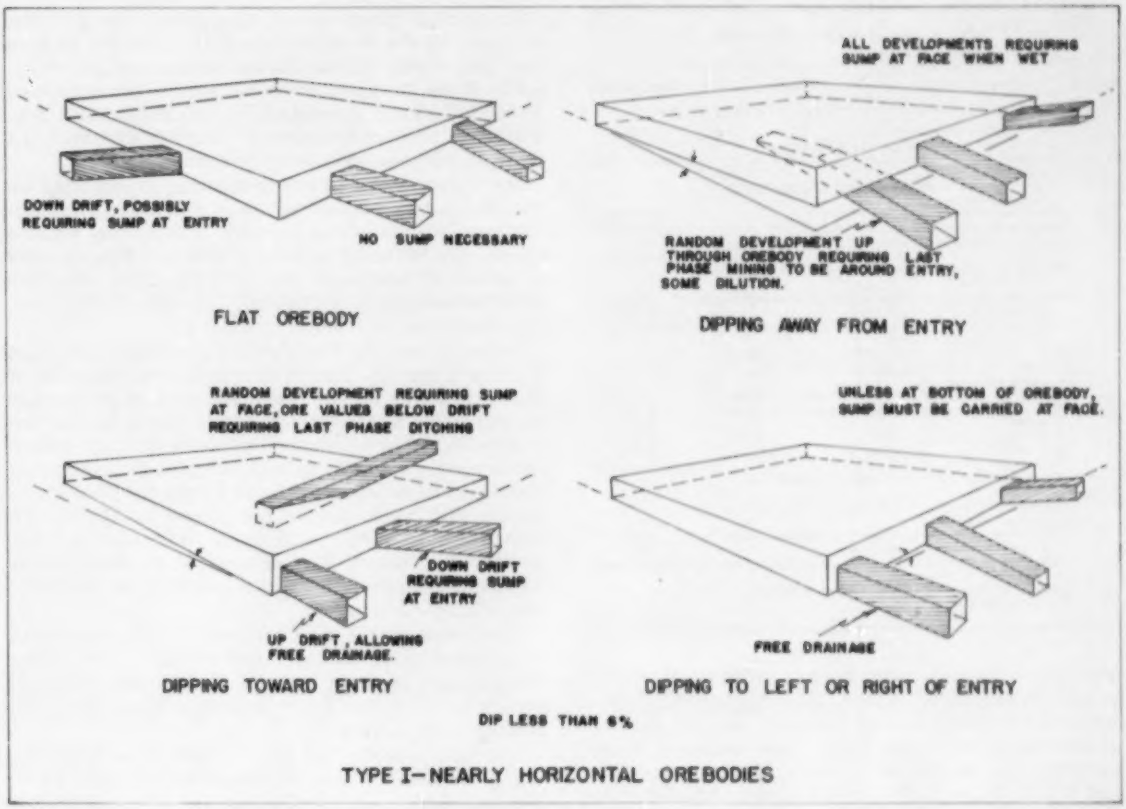


Fig. 11—Recommended development for type I orebody.

Fig. 12—Recommended development for types of orebodies.

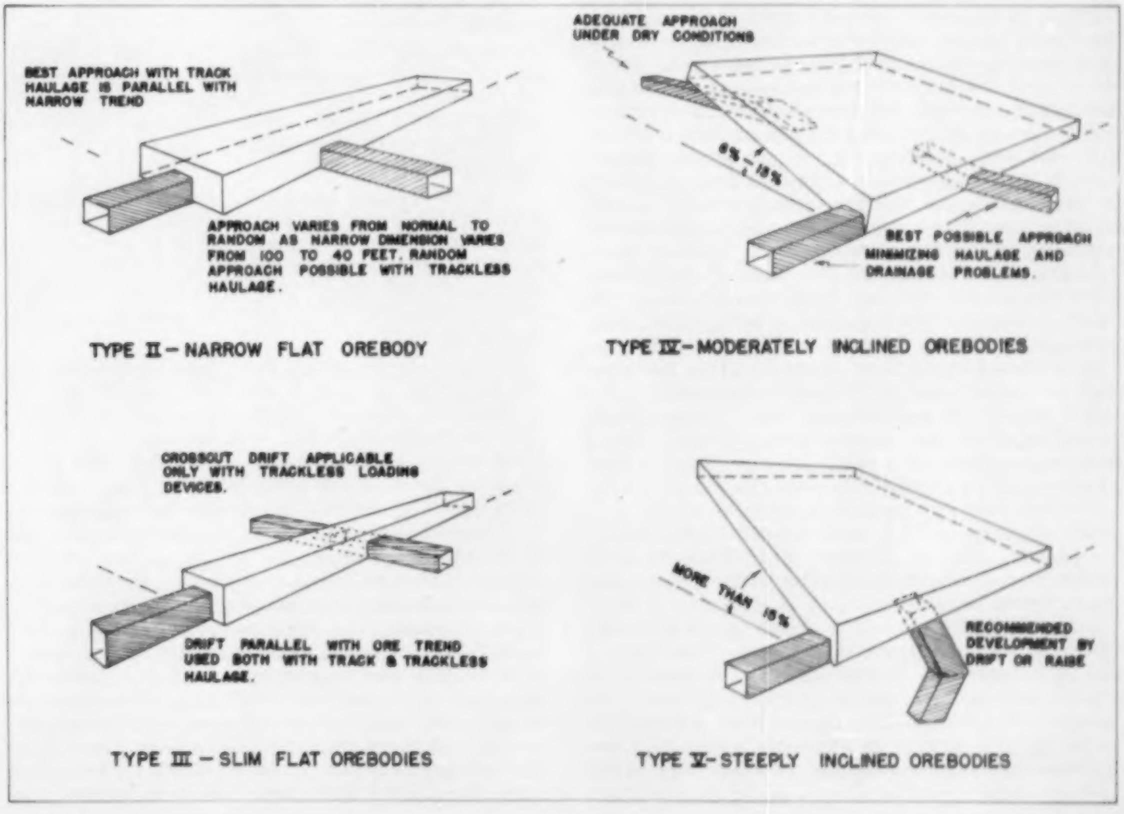


Table I. Rating Scale for Orebodies

Rating	Requirements	Must satisfy
1	Well located on structure. Return in excess of development cost. Horizon has good production history in area. Grade of ore well in excess of minimum requirements.	All four requirements
2	Good structural position. Return in excess of development cost. Horizon has good production history in area. Grade of ore in excess of minimum requirements.	First two requirements and one other
3	Fair structural position or insufficient information to merit a higher rating. Average chance of giving a return in excess of development cost. Past production history from ore horizon in area not necessary, but it cannot have been unfavorable. Grade of ore near minimum requirements.	All four requirements
4	Ore horizon has poor production history in area. Poor rock type. Off structure. Marginal grade. Development cost may equal or exceed return. Insufficient positive information to merit a higher rating.	Any two of the first four requirements
5	Ore horizon has thoroughly unsatisfactory production history in area. Ore occurs exclusively as fracture filling or as cubes in vugs. No. 4 orebodies disproved by additional prospecting.	Any one requirement

features of the host rock. Mining habit is not related to any stratigraphic zoning, although zones show a disposition toward one or toward a limited number of mining habits. The major zone contacts at all horizons may be similar in habit. If of sufficient size and tenor, an orebody within a reef complex may have the same mining habit as a bedded deposit.

Mining habit must be related to the basic methods and equipment used in the district. Mining methods in southeast Missouri are usually based on the St. Joe shovel and track haulage. Major variations in equipment include the Joy loader, the Eimco shovel, and the Rogers ramp for loading, the shuttle car and Diesel truck for short haulage, and various types of drags for scraping ore into chutes. Methods that influence classification of orebodies include track haulage, which is restricted to grades of 6 pct or less; shuttle car haulage, which cannot operate effectively on grades over 15 pct; and St. Joe shovel, which requires a minimum operating width of 20 ft.

Types of Orebodies: On the basis of floor slope, orebodies are grouped into three classes: those with nearly horizontal floor (under 6 pct); those with moderately dipping floor (6 to 15 pct); and those with steeply dipping floor (over 15 pct). Subdivisions are made according to width of orebody.

It is not always economically feasible to approach an orebody from the ideal position. Choice of entry must be in relation to available mine openings from which the development can start. In the following discussion drainage problems will be given only minor attention. They must be considered where the orebody lies at a lower level than adjacent workings and where the orebody dips away from the direction of approach.

Type I orebody has a floor which slopes less than 6 pct and a minimum width of 100 ft. This width is chosen because this is the minimum dimension at which any angle of approach will allow complete reversal of track within the orebody. With less width certain angles of approach necessitate redevelopment, leaving some ore, or taking low grade rock.

Orebodies falling within this group include most of those on the main stratigraphic contacts such as the 15/19 and 12/15. Many reef orebodies, especially those in which ore is not too closely restricted to individual roll structures, fit into this group. Orebodies of this category have thus far provided the bulk of southeast Missouri production.

As shown in Fig. 11, approaching an orebody on the low side presents the most attractive situation but may not always be possible. Frequently when a choice can be made a little additional development in order to approach the orebody from the best possible position is worthwhile to secure better mining conditions and lower mining cost.

Type II orebody has a floor that slopes less than 6 pct and has one lateral dimension less than 100 ft but more than 20 ft. The orebody is wide enough for operation of the St. Joe shovel but is too narrow to permit complete reversal of track haulage within the orebody. Unless the initial approach is correctly chosen some redevelopment is necessary in order to obtain maximum extraction. These orebodies are best suited to trackless haulage. With use of track, preferred mode of development is to make entry relative to shape and trend of orebody as shown in Fig. 12.

Orebodies of this type are common and provide a large amount of the ore now being mined. They include such geologic features as fan structures and on-lap zones in the lower beds and many reef orebodies in the 7 zone.

Type III orebody has a floor that slopes less than 6 pct and has one lateral dimension less than 20 ft. This orebody is too narrow to mine with the St. Joe shovel without excessive dilution of ore. Under present mining practice these orebodies are not mined, as the ore zone is too narrow to carry a heading.

Occasionally these orebodies are found in the 12 and 15 beds. In such cases they are extremely difficult to define. The most common occurrence is within the reef complex where individual roll-edges may be too widely separated to mine two or more roll-edges as a single heading.

Type IV orebody has a floor sloping more than 6 pct but less than 15 pct. As the orebody is too steeply inclined for track haulage width is not a factor. Orebodies of this type occur as bedded deposits at all stratigraphic zones. They are especially common in the 12 zone and on the flanks of sedimentary arch structures.

In those stopes which have been developed for track haulage mining is done in parallel benches resulting in considerable dilution of ore and multiple development to stay with the dip of the mineralization. Development of ore at the level indicated by the drillhole usually does not provide an entry favorable for mining. Attention must be given to slope of floor and trend of orebody prior to development.

Type V orebody has a floor sloping more than 15 pct; width is not a factor. Although not common, these orebodies have contributed considerable tonnage. They are found along the flanks of the structural centers and on porphyry knobs. Usually old stopes of this type have undergone several periods of redevelopment to get on the ore at different elevations, since ore goes into the back on one side of the working face and into the bottom on the other side. Invariably such stopes have produced low

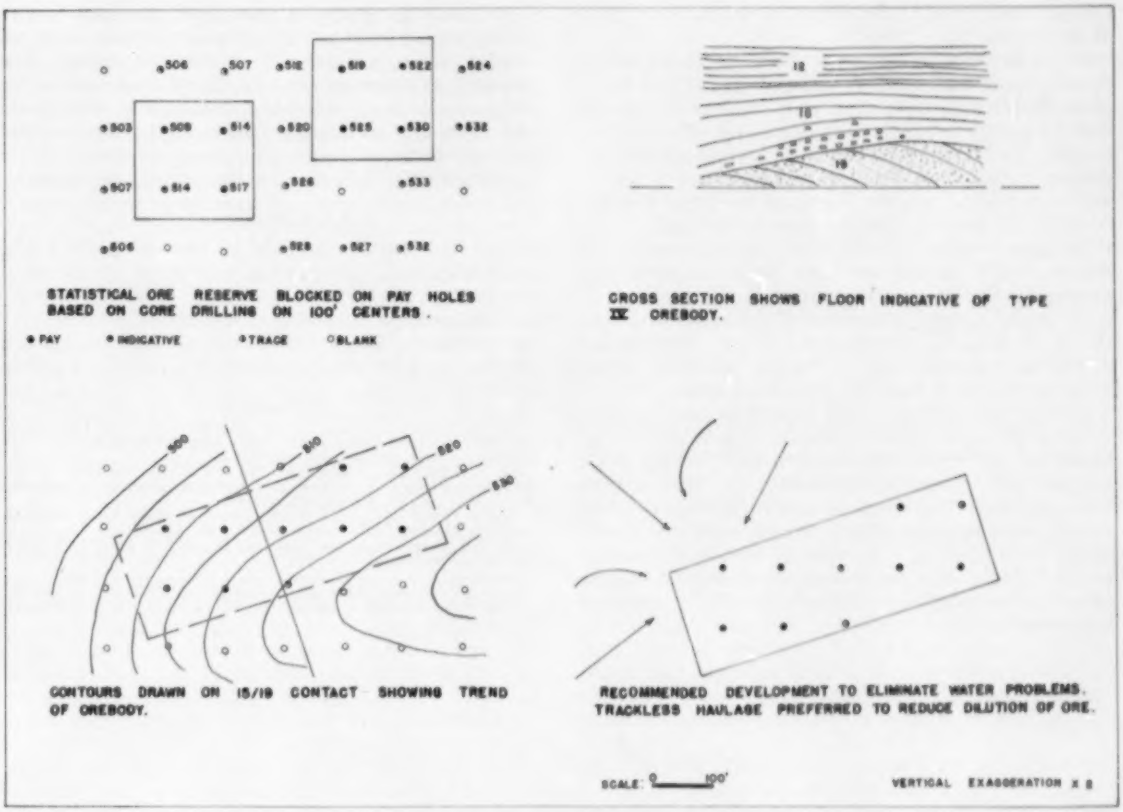


Fig. 13—Steps in evaluation of orebody.

grade ore because of the large amount of dilution necessary to maintain a mining floor.

Recommended procedure is to leave these orebodies until equipment is available which is adaptable to slope of floor. More favorable equipment, through decrease in both dilution of ore and redevelopment costs, makes these orebodies considerably more profitable to mine.

Several steps in evaluating a bedded orebody are shown in Fig. 13. The first illustration shows statistical ore reserves blocked on pay holes; elevations given indicate bottom of the ore in each hole. Contours on the 15/19 contact which carries the ore are shown on the second illustration. A cross section of the orebody, shown in the third illustration, reveals the nature of the mining floor, which determines the type of orebody. Recommended development is shown in the fourth illustration.

Geologic Aids to Mining

Periodic Stope Visits: Only by frequent and regular contact with the operating staff can the geologist become familiar with the problems that face members of the mining department in their efforts to maintain grade and tonnage and to control mining costs. Such contacts cement relations between geological and operating staffs and develop a cooperative approach to mutual problems. An effort is made to acquaint the operating staff with major aspects of geology of the mine. The main zone contacts, recognition of rock type in each zone, important types of structures that control ore trends, and recognition of displacement on faults represent useful information to the operator. The geologist gains much practical knowledge of ore based on experience of operating staff and miners, as well as becoming familiar with operating problems important in evaluation of orebodies.

Anticipation of trends of minor structures that control ore is important. Lack of attention to dip of contact may result in mining of excessive amounts of waste rock or even loss of ore in high breast or bottom. In mining downhill miners tend to raise the bottom to get out of water and may leave good ore in the bottom. Close attention to structural trends reduces the amount of redevelopment necessary for maximum extraction.

Visits to operating stopes are also useful in planning prospecting. Jackhammer prospecting by the operating driller is used to explore potential ore-bearing structures revealed in mining. A considerable amount of ore not outlined by the diamond drilling that led to initial development of the orebody is found by this method. Structures not accessible to jackhammer prospecting are recorded as targets for future diamond drilling.

Stope Evaluation: Orebodies being mined and those developed and ready to mine are evaluated each quarter to provide an estimate of the productive life of the stope and to anticipate development and prospecting needs. Tonnage of ore remaining is estimated on the basis of knowledge of the orebody from direct observation, structural position, and drill information. Quality and extent of the information are considered. Estimated tonnage of unmined ore is converted to number of drill shifts according to the average rate of breaking per drillman per shift at the mine concerned. This indicates

the probable life of the stope in months or years of operation.

Stope evaluation data are useful in coordinating development and mining. Mining practice is based on a shovel unit consisting of a St. Joe shovel and four to six operating drillers breaking rock for that shovel. To eliminate unnecessary moving time, operators attempt to keep the stopes that a shovel serves restricted to a small area of the mine. Knowledge of the future of each stope in the unit facilitates preservation of the unit. Replacements for stopes nearly mined out can be anticipated and nearby orebodies can be prepared for mining. At most mines a flexible balance is maintained among stopes in different stratigraphic zones. Knowledge of the approximate life of stopes indicates which levels of the mine need further development.

The objective is to avoid expedient development at the expense of future operation and to give better balance to the mine which may lead to a longer life and more complete ore extraction. A basis is provided for deciding whether attention can be given to long development jobs or whether efforts should be made to open up a number of orebodies quickly, whether high tonnage, low grade orebodies can be developed or whether attention should be given to improving grade.

Information given in the stope evaluation also points up additional development that may be needed before all the orebody can be mined. The development necessary to continued operation of the stope can be anticipated and worked into the schedule. Geologic knowledge that is useful in anticipation of future mining problems in the stope is brought to the attention of the mining department.

Acknowledgments

The writers have profited by discussing the problems of geology and mining with many members of the geological and operating staff. Their many useful suggestions and criticisms are gratefully acknowledged. The authors are indebted to the St. Joseph Lead Co. for permission to publish the paper.

References

[1] E. R. Buckley: *Geology of the Disseminated Lead Deposits of St. Francois and Washington Counties*. Missouri Bureau of Geology and Mines, 1908, vol. 9, parts 1 and 2.

[2] E. L. Ohle and J. S. Brown (ed.): Geologic Problems in the Southeast Missouri Lead District. Geological Society of America *Bull.*, 1954, vol. 65, pp. 201-222, 935-936.

[3] W. A. Tarr: Origin of the Southeast Missouri Lead Deposits. *Economic Geology*, 1936, vol. 31, pp. 712-754, 832-866.

[4] R. E. Wagner: Lead Belt Geology. *Mining and Metallurgy*, 1947, vol. 28, pp. 366-368.

Discussion of this paper sent (2 copies) to AIME before Feb. 28, 1957 will be published in MINING ENGINEERING and in AIME *Transactions*, Vol. 208.

Improved System Measures Heavy Slurry Density

by Bernard Rachlin

IN the mineral industry measurement of density or specific gravity of slurries is often necessary or highly desirable. To date the most successful method of measuring the specific gravity of various media is to hand weigh a constant volume sample of the slurry. This method is time-consuming, however, and is intermittent rather than continuous.

Most density measuring instruments in use today are variations of three basic systems: air bubble tube systems, radioactive methods, and displacer systems.

The high cost of the radioactive cell, combined with the precision electronic measuring that is required, limits the practical utility of this procedure to a small percentage of critical processes. In the air bubble systems clogging of the bubble tubes with solids has rendered this type of measurement completely ineffectual for many applications. Continuous or intermittent purging of the bubble tubes with air or water to prevent tube clogging has been tried but has not been very successful, especially in the heavy media process.

The displacer system, which presents the fewest complications of all the density measuring systems, consists only of a metal immersion unit and a means of weighing it. In the past, settling of the solid in the slurry has caused untrue samples and clogging of the media at the bottom of the chamber. It is also important to dampen excessive flow surges such as those caused by reciprocating pumps. The system described here successfully overcomes these difficulties.

Description of the Process System: The process equipment, on which the field tests were run, was a

B. RACHLIN is an Application Engineer, Application Engr. Div., Minneapolis-Honeywell Regulator Co., Philadelphia.

TP 4375B. Manuscript, March 1, 1956. New York Meeting, February 1956.

Dorrco Fluo-Solids system, Fig. 1, used in roasting sulfides for sulfur dioxide production. For best efficiency it is necessary that a heavy slurry of pyrite ore and water with specific gravity between 2.60 and 2.65 be pumped into the reactor.

Location of the density measuring system in the process was of very great importance. For proper control of density, it was necessary that density of the ore slurry be measured as soon as possible after the pyrite ore and water were mixed. Also, the density system had to be located so that advantage could be taken of a process pump to circulate the slurry through the sampling chamber.

With these two points in mind, it was decided that the density system should be located at the end of the pipeline from the make-up tank to the top of the storage tank. At this location the reciprocating pump would pump the slurry from the make-up tank, through the sampling chamber, and to the storage tank. Thus the density of the slurry would be measured only a few minutes after the slurry was made.

Type of Density Measuring System: This density measuring system consists of a detecting unit and a transmitting unit. It produces a transmitted pressure, with a range of 3 to 15 psi, that is proportional to the density of the slurry in the sampling chamber. An indicating, recording, or controlling instrument calibrated for the 3 to 15 psi range can be connected to the transmitter. More than one receiving instrument can be connected if desired.

The detecting unit in this case is a displacer suspended in the slurry in the sampling chamber. A density change is detected by the apparent change in weight of the displacer, which results from the changed buoyant force of the slurry on the displacer. The change in apparent weight of the displacer, caused by the change in specific gravity of the slurry, is then transferred to the transmitting unit.

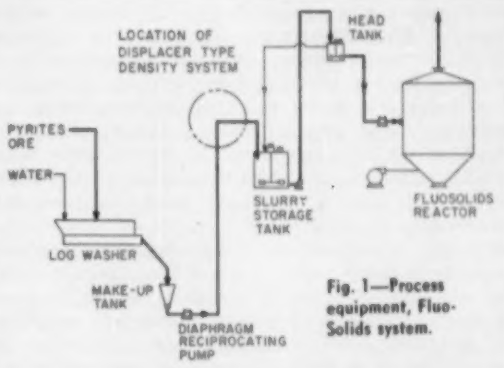


Fig. 1—Process equipment, Fluo-Solids system.

The relationship between the apparent weight of the displacer and the specific gravity of the slurry is as follows (Eq. 1):

$$A.W. = T.W. - 62.4(V)(S.G.) \qquad [1]$$

where $A.W.$ = apparent weight of displacer in pounds = $T.W. - B.F.$, $T.W.$ = true weight of displacer in pounds, $B.F.$ = buoyant force of the slurry in pounds = $62.4\ (V)(S.G.)$, V = volume of displacer in cubic feet, and $S.\ G.$ = specific gravity of the slurry. Density of water under standard conditions of temperature and pressure = 62.4 lb per cu ft.

Since $T.W.$ and V are constants for any given installation, the apparent weight must vary inversely with specific gravity. The displacer was so designed that its actual or true weight would always be heavier than the weight of the volume of slurry displaced in the sampling chamber.

The Brown pneumatically operated differential

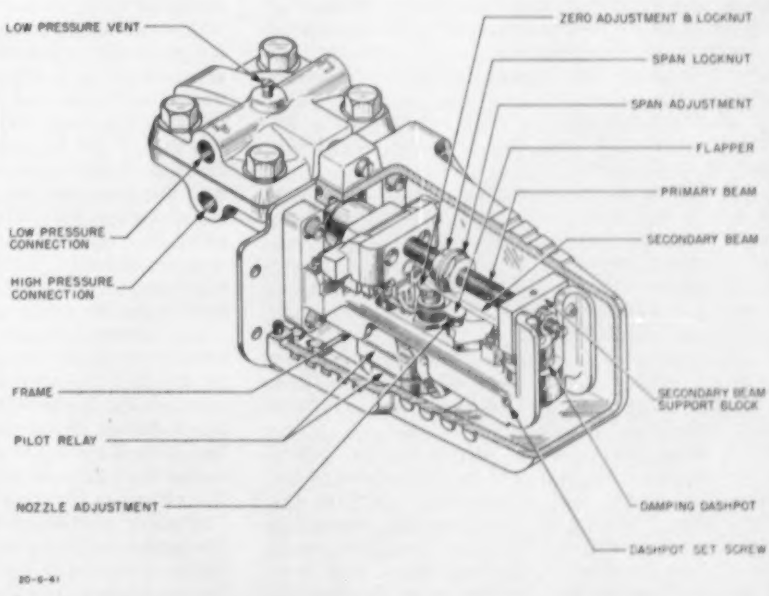


Fig. 2—Cutaway view of differential converter with pilot relay.

converter unit (see Fig. 2) is an ideal instrument for measuring this apparent change in weight of the displacer. The differential converter unit works on the pneumatic balance principle, wherein a force resulting from a change in the variable is opposed by a pneumatic force, and this opposing force is a measure of the variable. The apparent change in weight of the displacer would be the variable. Since forces are balanced rather than motion, movement of the unit is held to a very small value—a few thousandths of an inch.

By pin-connecting and suspending the displacer from the primary beam of the differential converter unit, a different pneumatic signal can be obtained for every change (or apparent change) in weight of the displacer. This pneumatic signal can then be transmitted to a Brown pneumatic receiver which continuously indicates and records these changes in units of specific gravity on a circular chart. For these tests a differential converter was calibrated to produce a linear pneumatic signal that would correspond to a range in specific gravity from 2.00 to 4.00, with an accuracy and readability to better than 0.02 sp gr.

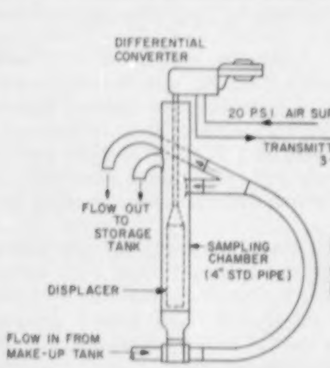


Fig. 3—Initial density system installation.

Design Requirements of the Sampling Chamber: The sampling chamber is one of the most important parts of this density measuring system. Because of the type of slurry being measured and because of the location of the chamber in the process, the sampling chamber must meet the following design requirements:

1) It must maintain a level of slurry in the chamber which will keep the displacer completely immersed at all times.

2) It must provide a sample of slurry to be measured which will truly represent the slurry being pumped from the make-up tank to the storage tank.

3) It must maintain a circulation of slurry through the chamber so that the density system can measure immediate changes in density of slurry from the make-up tank.

4) It must not clog at the bottom of the chamber because of settling out of the solid material. (This has been the biggest stumbling block to all types of density measuring systems in the past.)

5) It must eliminate the surges due to normal flow and, in this installation, to the pulsations of the reciprocating pump that pumps the slurry to the storage tank. Pumping of this type can cause the most violent flow surges throughout the entire process as well as the density chamber, but it may present no problem for the remainder of the system. Because flow surges do present such a problem in the displacer-type system, some means of dampening these surges must be found before this density measuring system can be acceptable.

The immediate problem was to design a chamber that would maintain the slurry at or above a certain level and simultaneously prevent clogging at the bottom of the chamber due to settling out of the solid material. Usual designs called for the slurry to enter at the top of the chamber and to discharge through the bottom of the tank. To obtain the necessary level to immerse the displacer completely, some type of restriction would be placed at the bottom. Theoretically, the restriction would permit a steady draining of the slurry entering the chamber, but not in sufficient volume to prevent the necessary level from being maintained. In actual practice this was not the case. If the restriction was too large the level could not be maintained, if too small the restriction plugged. A number of adjustable restrictions such as pinch valves were tried, but none proved very successful.

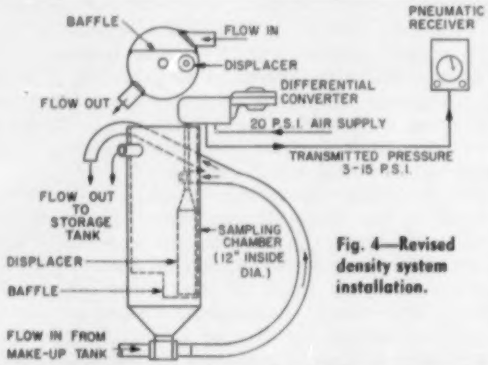


Fig. 4—Revised density system installation.

Initial Design of the Sampling Chamber: Since the usual chamber designs would not be satisfactory for slurries of the type used in the Fluo-Solids system for roasting sulfides, another line of attack had to be taken. The initial design, Fig. 3, called for the chamber to be made from a 4-in. standard steel pipe 3 ft long. The bottom of the chamber was reduced for assembly with a 2-in. T pipe fitting, and the pipeline from the make-up tank was union-connected to one end of the fitting. A hose connection, fitted to the other end of the T, was swept up in an arc to a Y fitting located approximately 1 ft from the top of the chamber. One branch of the Y fed into the chamber, the other branch directly into the main storage tank. An overflow line, from a fitting assembled about 9 in. from the top of the chamber, also fed directly into the main storage tank. There were no other pipelines feeding the main storage tank from the make-up tank.

The theory behind this design was that constant circulation could be maintained inside the density chamber. Flow would be from the main pipeline, through the T connection, and in a sweeping arc to the Y fitting. From the Y, part of the slurry would flow into the sampling chamber and the remainder would flow directly to the main storage tank. Inside the chamber the flow would be in two directions. The major portion of the slurry would flow through the overflow fitting to the main storage tank. The minor flow would be down through the bottom of the chamber to the T and then swept into the main slurry flow from the make-up tank.

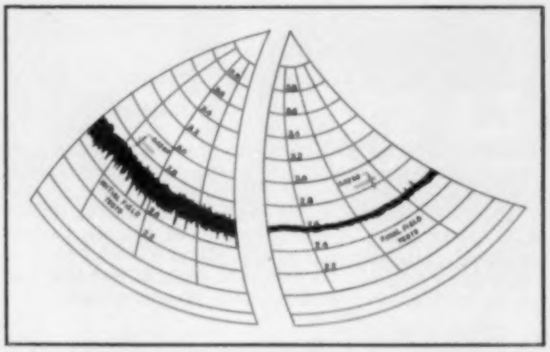


Fig. 5—Comparison of initial and final chart records, showing extent of reduction.

Clogging at the bottom of the chamber could not occur because any solid material that might tend to settle out would be swept into the slurry flowing through the T assembled to the main pipeline. Since the slurry could only be discharged from the density chamber through the overflow fitting, the level in the chamber would always be maintained at or above this point. Since the top of the displacer was located at a point about 6 in. below the overflow fitting the displacer would always be fully immersed.

Results of Initial Field Tests of Density Measuring System: A density measuring system was assembled, incorporating the designs of the modified differential converter and sampling chamber as previously described. The displacer, made of stainless steel, was a cylinder of 3 in. diam and 17 in. long. A stainless steel rod ¾ in. diam was welded to the top of the displacer and pin-connected to the end of the primary beam of the differential converter. With air at 20 psi supplied to the differential converter and with a pneumatic receiver, calibrated to indicate and record the specific gravity of the slurry through a range from 2.00 to 4.00, pneumatically connected to the output of the differential converter, the displacer type density measurement was placed in operation.

Results of the initial tests were very gratifying but not completely successful. The primary problem that caused all previous attempts of continuously measuring slurry density to fail was solved. In more than two months of process operation, and for a period of about 5 hr each day when the ore slurry was being made and pumped to the main storage tank, the displacer remained completely immersed while the necessary circulation was maintained to prevent clogging at the bottom of the sampling chamber. Not a single instance of clogging occurred during this period of operation.

The unsuccessful portion of the test was due to the violent flow surges through the sampling chamber produced by the pulsations of the reciprocating pump. From the chart records, Fig. 5, it seemed that the density was cycling through a band 0.20 *S.G.* wide. Because of the erratic pulsations of the pump it was impossible to determine even approximately the true specific gravity point between the two extremes. In addition, due to the wide cycling band, slight changes in density of the slurry did not show up in the chart record.

Revised Design and Field Tests: It was apparent that some means must be found to dampen surges in the sampling chamber. Since the primary purpose of the sampling chamber had been accomplished, it was necessary that the same general arrangement of the original design be maintained. Through laboratory tests of various modifications of the original design the present design was evolved. It is shown diagrammatically in Fig. 4. A specially designed baffle was added to direct the flow of slurry through the chamber.

The redesign has practically eliminated the surges in the slurry surrounding the displacer. Fig. 5, which compares initial and final chart records, shows the extent of the surge reduction. Whereas the original installation cycled through a band 0.20 *S.G.* wide, the maximum band through which the latest installation cycles is 0.04 *S.G.* Samples of media taken from the chamber at the displacer and manually weighed indicated that the true specific gravity could be considered the midpoint of the 0.04 *S.G.* band. In the latest tests, therefore, the density measuring system measured to within 0.02 of the actual specific gravity.

This system has also shown accurate and almost immediate response to all density changes in the media. Samples of media taken from the sampling chamber were compared to samples of media flowing directly to the main storage tank. The same specific gravity was found for comparable samples.

After one month of operation this density system continued to perform very satisfactorily. No build-up of solids at the bottom of the chamber resulted, and no special attention or maintenance procedure was required to keep the system operating. After each day's operation, to prevent accumulation of dried slurry in the pipeline, clear water is pumped for about 5 min. through the pipeline between the make-up and storage tanks. Any excess water left in the pipe is then drained through the pump. No alteration to this procedure was necessary with the installation of the density system. The sampling chamber is cleaned and drained along with the pipeline.

Conclusions

Successful development of a displacer-type density measuring system for slurries has been accomplished. The differential converter unit, modified for assembly with the displacer, proved very satisfactory in measuring the slurry density in the sampling chamber and transmitting it to the Brown pneumatic receiver recorder.

Design of the sampling chamber, the key to the entire density system, maintained the level of a true representative sample of slurry to keep the displacer completely immersed. No clogging or even build-up resulted at the bottom of the chamber due to the settling out of the solid material. Violent surges caused by the reciprocating pump which pumped the slurry from the make-up tank to the main storage tank were eliminated. Necessary circulation was maintained so that slight changes in slurry density from the make-up tank were immediately sensed by the displacer and indicated by the pneumatic receiver.

This design should be applicable to any slurry density measurement where a sample can be pumped by any means to the sampling chamber. With such a sensitive, stable, and trouble-free measuring system, addition of conventional control systems to maintain constant specific gravities can readily be accomplished.

Discussion of this paper sent (2 copies) to AIME before Feb. 28, 1957, will be published in MINING ENGINEERING and in *AIME Transactions*, Vol. 208.

Discussion

Mineralizing Solutions That Carry and Deposit Iron and Sulfur

(MINING ENGINEERING, page 1012, October 1956; *AIME Trans.*, vol. 205)

by Eldred D. Wilson

Apropos of metals low in the series illustrated by Dr. Butler's Table II, it is interesting to note that Cress and Feldman[29] reported traces of platinum metals in several samples of alunite. They found, chemically and spectrographically, several traces of platinum, palladium, and rhodium in alunite from Sugar Loaf Peak, near Quartzite, Ariz.; silver, platinum, and rhodium in alunite from Rio Grande County, Colorado; and traces of rubidium and gallium in various samples of alunite. These authors stated that the platinum metals appear to be present as oxides and suggested deposition by hydrothermal solutions under near-surface conditions.

As described by Heineman,[30] the wall rock of the Sugar Loaf alunite veins shows considerable alunitic alteration. Pyrite and hematite are relatively abundant in the wall rock but decrease progressively near the veins.

Assays of samples of the Sugar Loaf alunite, run by J. B. Cunningham of the University of Arizona College of Mines, showed traces of silver and 0.002 to 0.004 oz of gold per ton, but the chemical form of the gold and silver in this occurrence remains undetermined.

References

[29] S. H. Cress and C. Feldman: Platinum Minerals Identified in Western Alunite. *Engineering and Mining Journal*, 1943, vol. 144, No. 12, p. 106.

[30] R. E. S. Heineman: Sugarloaf Butte Alunite. *Engineering and Mining Journal*, 1935, vol. 136, No. 3, pp. 138-139.

Discussion

Analysis of Roof Bolting Systems Based on Model Studies

(*Mining Engineering, page 954, October 1956; AIME Trans., Vol. 202*)

by J. P. Zannaras

If we assume that testing of the model started at time t_1, that time t_2 was the instant at which the elastic limit of the material was passed at the points of the maximum stress, and that at time t_3 the failure of the model was completed as shown in the typical photograph (Fig. 1), then actually the photograph shows the final results of what took place between time t_2 and t_3.

By the principle of similitude all conclusions drawn from the behavior of the model were applicable and valid for actual mine roofs up to the time t_2 after time t_2 motions and friction came into play during the destruction of the model.

The events between t_2 and t_3 and the photograph itself are not representative of the events expected in an actual mine roof unless corrections imposed by the principle of similitude are made.

It can therefore be stated that all conclusions and observations of the author from the photographs are applicable only to his small-scale experiments and not to actual mine roofs.

Without investigation of the true stresses or the combined maximum stress induced in the beds it has been assumed by the author that failure occurred solely due to bending stress caused by loading of the beds. (The author states that *D* was taken to be the theoretical strain given by elementary beam theory.)

Fig. 1 shows that the beds were clamped together. This clamping was necessary to prevent slipping of the beds due to the horizontal shear. However, this clamping caused an uncontrolled and undertermined compressive stress *S* at the end of the beds, and this compressive stress caused a true tensile stress λS (λ Poisson ratio).

This true tensile stress combined with the tensile stress produced at the end of the beds due to loading (fixed beam uniformly loaded), and the horizontal shear which is maximum at the end appears to be the most probable maximum stress produced in the beds.

It is therefore evident that the author in performing this experiment has introduced stresses in the model not existing or dissimilar to actual mine roofs, and therefore his conclusions may be applicable to his small-scale experiments but inapplicable to actual mine roofs.

Louis A. Panek *(author's reply)*—The discusser's difficulty is related to mechanics of materials rather than similitude. The effect of time did not enter into the tests in any way, being simply excluded from consideration in this study, because all quantitative results entering into the design equations are based on measured (between time t_1 and time t_2) bending strains that are well below the breaking strain (i.e., in the elastic range) for the model material. Time does not become a significant factor until the rock stress reaches 80 to 90 pct of the breaking stress.[4] The objective of this investigation was not to predict that bolted roof will *fail* after time t_2 but was instead to develop a *safe* roof, that is, roof in which the bending strain is much less than (say less than 50 pct of) the breaking strain.

The models tested to failure had no direct bearing on the design equations. Although for some rocks time may influence the magnitude of stress or strain at which fracture occurs, breaking stress or breaking strain values were not employed for any purpose in this investigation. Tests to failure were made only to obtain additional evidence regarding the behavior of the bolted laminae[5] as evinced by the locations of the cracks. As stated in the last paragraph of the paper, if one wishes to predict the time or stress conditions for which the roof will fail, additional information is required.

The reason for clamping the laminae is to make them behave liked clamped beams, which closely approximate the clamped-plate condition of the actual mine roof beds (Ref. 5, p. 1; Ref. 6). The writer has previously demonstrated that the bending stresses induced by centrifugal loading in a clamped model beam are in agreement with those predicted by theory;[7] hence clamping has no significant effect on the bending strains measured during a test. Moreover, actual mine roof beds are likewise subject to a clamping effect due to weight of superincumbent strata carried by the pillars or ribs on each side of the opening.

It is emphasized that this study was restricted to evaluating only the reinforcing effect produced by the bolts. Procedures employed in the experiments and in the data analysis were such as to deliberately exclude the effects of other factors. The objections offered are therefore invalid.

References

[4] F. D. Wright and P. B. Bucky: Determination of Room and Pillar Dimensions for the Oil-Shale Mine at Rifle, Colorado. *AIME Trans.*, 1949, vol. 181, Fig. 7.

[5] L. A. Panek: Principles of Reinforcing Bedded Mine Roof With Bolts. *USBM R. I. 5156*, 1956, p. 19.

[6] L. A. Panek: Theory of Model Testing as Applied to Roof Bolting. *USBM R. I. 5154*, 1956, pp. 9-10.

[7] L. A. Panek: Centrifugal Testing Apparatus for Mine Structure Stress Analysis. *USBM R. I. 4883*, 1952, p. 17.

1957 Annual Meeting Mining Branch Technical Program Takes Shape

(Data as of Nov. 1, 1956)

As the dates for the 1957 Annual Meeting draw close, Program Chairmen are checking final details of the Mining Branch technical fare. Mining sessions will be at the Roosevelt Hotel, and final times and dates for sessions will appear in the complete program in the January issue of MINING ENGINEERING.

Minerals Beneficiation

Pyrolysis and Agglomeration

Robert E. Powers, Session Chairman: *Down-Draft Pellet Hardening*, by Alan English and M. F. Morgan; *Up-Draft Pellet Hardening*, by L. J. Erck; *Nodulizing*, by Morris Mielke and R. E. Hagen; *Lurgi Pelletizing Process—A Combination of Down and Up-Draft Operation*, by Kurt Meyer; *Pelletizing in a Shaft Furnace*, by F. D. DeVaney.

Crushing and Grinding

D. J. Drinkwater and C. Beech, Session Chairmen: *Comminution Exposure Constant by the Third Theory*, by Fred C. Bond; *Rod Mill Experience in Grinding Magnetic Taconite Ore at Reserve Mining Co.*, by A. S. Henderson and E. M. Furness; *Energy-Size Reduction Relationships in Comminution*, by R. J. Charles; *Grinding Practice at Tennessee Copper Co.'s Isabella Mill* by F. M. Lewis.

Mill Design

O. W. Walvoord and W. L. Howes, Session Chairmen: *Soil Studies for Foundations*, by E. H. Bronson; *Electrical Mill Design*, by C. B. Risler and W. E. Thomas; *Mill Operating Records and Accounts*, by Nathaniel Herz; *Factors in Vibrating Screen Installations*, by N. Kuenhold.

Symposium

Cyclones for Wet Classification, R. H. Lowe, Session Chairman; *Development of the Hydro-Cyclone*, by Stephen E. Erickson; *Compilation of Cyclone Operating Data*, by T. M. Morris; *The Use of Cyclones in the Grinding of Taconite*, by F. D. DeVaney; *Factors Influencing the Choice of a Cyclone for Wet Classification*, by E. C. Herkenhoff; *Cyclone Practice in Arizona*, by Russell Salter and Edwin J. King.

(Continued on page 1233)

1957 ANNUAL MEETING
MINING BRANCH SOCIAL FUNCTIONS

SUNDAY, FEBRUARY 24

6:30 P.M.	Mineral Industry Education Division Dinner

MONDAY, FEBRUARY 25

12:00 Noon	Welcoming Luncheon
6:00 P.M.	Cocktail Party
7:30 P.M.	Dinner-Smoker, Stag Dinner and entertainment

TUESDAY, FEBRUARY 26

7:30 A.M.	Minerals Beneficiation Div. "Scotch Breakfast"
12:00 Noon	Mining, Geology, and Geophysics Div. Luncheon Coal Div. Luncheon
6:00 P.M.	Cocktail Party
7:00 P.M.	Mining Branch Annual Dinner
10:00 P.M. to 2 A.M.	Informal Dance—Moonlight Sail aboard S. S. President

WEDNESDAY, FEBRUARY 27

12:00 Noon	Industrial Minerals Div. Luncheon
12:00 Noon	Mineral Economics Div. Luncheon
7:00 P.M.	Annual AIME Banquet and President's Reception

THURSDAY, FEBRUARY 28

12:00 Noon	Minerals Beneficiation Div. Annual Luncheon

Surprises in Store at Annual Meeting

Highlighting the social program planned for the Annual Meeting in New Orleans, is the Informal Dance and moonlight sail aboard the *S.S. President*, on Tuesday February 26. The floor show, scheduled for midnight, will feature a comic magician MC, a singing duo and a dance team. In addition to accordian and guitar music, an eight-piece orchestra will provide dance music for the guests. Soft drinks and beer will be sold, but tickets read, "Bring your own liquor."

The traditional Welcoming Luncheon is set for 12 noon, February 25, at the International Room of the Roosevelt. After an address by the guest speaker, awards will be presented to winners of local section membership and student prize paper contests. Souvenirs will be furnished by the Freeport Sulphur Co.

The International and Grand Ballrooms of the Roosevelt Hotel will accommodate the 1600 guests attending the Annual Banquet on Wednesday, February 27 at 7 pm. Prior to dinner, a reception will be held in the Wildcatter Room for Head Table guests. During the banquet, background music will be played by the Bert Peck String Ensemble, with entertainment commencing at midnight.

"For men only, the dinner-smoker at the Roosevelt on Monday, February 25, at 7:30 pm, will feature "excellent stag entertainment."

Early risers the next morning can join the famous MBD Scotch Breakfast replete with appropriate entertainment and food.

Two Outstanding Local Sections Hold Meetings With Varied Social Activities, Interesting Plant Visits

Black Hills Section Tours Uranium Mines and Mills

The Black Hills Section of AIME held a regular meeting Saturday, September 22nd, with sessions at Edgemont, S. D., and the Sylvan Lake Resort near Custer, S. D.

Highlighting the all-day session was a trip through the Mines Development Inc., uranium processing mill at Edgemont, and a tour of the Gould and Kellogg uranium producing properties near Edgemont. The day's affair began with a luncheon at 11:30 am at the Stockman's Cafe Dining Hall in Edgemont served to 55 members and their wives, student associates of AIME from the School of Mines and Technology in Rapid City, and operating and management personnel from the Mines Development Inc. uranium mill.

Following the luncheon, preview tours of the mill and uranium field trips were presented by personnel from Mines Development Inc. and the AEC. J. T. Vogenthaler, planning engineer, Mines Development Inc., gave a general talk covering the background of the company, comments on the mill itself, the use of local labor as much as possible for mill operation, the recreation and employment benefit programs instituted for employees, and the relationship of Mines Development with the city and residents of Edgemont and the AEC in the operation of the uranium mill. H. L. Hazen, mill superintendent, presented features which governed the design and construction of the Edgemont mill sampling procedures; dust control; automatic control equipment incorporated in the mill for operation and control; and percentage recovery figures of the mill.

Ray Bartley, AEC district geologist, presented the preview tour of the uranium fields giving the geology of the areas, and mineralogy of the two producing properties to be visited by the group. He also conducted the group through the Gould and Kellogg properties following the tour of the uranium mill.

At 6:30 pm a buffet style dinner was served to 60 AIME members and their wives, student associates, and Mines Development personnel in the Sylvan Lake Resort Dining Hall. A regular business meeting and program followed the dinner. The business portion of the meeting included voting on an amendment to the by-laws, and discussion of the meeting in Rapid City on October 16th.

Official guests of the Black Hills Section meeting were AIME Mining Branch Secretary Arnold Buzzalini and Rixford Beals, editorial director of MINING ENGINEERING and JOURNAL OF METALS. Both Mr. Buzzalini and Mr. Beals were called upon to give short talks at the evening session at Sylvan Lake, and enlightened the group on matters pertaining to Institute affairs.

Principal speaker for the evening was John King of the AEC in Rapid City, who chose for his topic *Summary of Recent Uranium Developments in the Western Dakotas.* Mr. King informed the group that uranium deposits are scattered across the Western Dakotas and into the adjacent states in a belt about 100 miles wide, extending from Edgemont, S. D. at the southern tip of the Black Hills, to west central North Dakota. He then briefly summarized the history of uranium in the Black Hills, and concluded with a discussion of lignite in the Dakotas and Montana.

When the skid-shovel emerged from the mine with a bucket full of ore, visitors picked samples to take home as souvenirs.

This photo of the tour of the Gould and Kellogg uranium mine shows Arnold Buzzalini, Mining Branch Secretary, left foreground.

Before actually visiting the mine, members were briefed by an AEC advisor who presented a quick run-down of the property and geology.

Loaded shovel is shown at mine entrance.

Successful Conclave Of Upper Peninsula Section Featured Tours, Parties

The most successful, best attended conclave of the **Upper Peninsula Section** took place at their Annual Meeting on October 6, in Ishpeming, Michigan.

Stress on Safety

The gathering of 245 members, including the auxiliary, met in the morning at Mather Inn. By 10:30 am they were strolling through town, replete with hard hats and safety goggles, in compliance with the stress on safety emphasized by program chairman D. Kelly Campbell.

Tour Republic Mine

By 11 am the train got under way for Republic's new mine. On the way Mr. Campbell handled introductions and pointed out the mines visible from the tracks. Twelve groups were formed when the members reached their destination, and each group was taken on a conducted tour of the open pit operations and the concentrating plant.

Visit Eagle Mills Plant

They stopped to pick up "pasties" for the trip to Eagle Mills, and arrived at the big pelletizing plant at 2:15 pm. After inspecting this plant, owned by Marquette Iron Mining Co., the group moved on to Presque Isle dock, stopping for a scenic view of Dead River gorge. The party arrived back in Ishpeming at 5 pm, in time for cocktails, followed by a business meeting-dinner.

Officers Elected

During the business meeting, members discussed the program for the coming year in the three subsections, Exploration, Mining, and Beneficiation. In the elections which were held, Burton Boyum stepped up from the vice chairmanship to replace Richard Moe as chairman of the section, and Richard Brewer became chairman-elect. DeRoy Dreyer was chosen secretary-treasurer again, and Messrs. Boyum and Brewer were selected for two-year terms to the national meeting. Mrs. Carleton Bailey succeeds Mrs. Boyum as chairman of the auxiliary, and Mrs. John Wangaard was elected secretary-treasurer.

AIME President Attends

The group was honored by the presence of AIME President Carl E. Reistle, Jr., and President-Elect Grover J. Holt.

Candid views of the Ishpeming gathering follow Section members from registration, to their excursion trip to Republic Mine, safety demonstrations at Eagle Mills, and finally to their wind-up dinner at Mather Inn.

AIME Awards

The following honors will be conferred at the annual banquet on Wednesday, February 27, 1957 at the Roosevelt Hotel, New Orleans.

Charles F. Rand Medal to **Andrew Fletcher,** "For distinguished administration in nonferrous mining and metallurgical enterprises; for his inspiring leadership in the mineral industry; and for his outstanding contributions toward furthering the professional interests of those engaged in the mineral industries."

James Douglas Medal to **Russel B. Caples,** "For distinguished achievement in extractive metallurgy, particularly in the electrowinning of zinc; and for outstanding success in stimulating the professional careers of many metallurgists."

Anthony F. Lucas Medal to **J. E. Brantly,** "For contributions to the art of drilling; for technical writing including the *Rotary Drilling Handbook;* and for leadership in the petroleum industry, notably in founding the American Association of Oil Well Drilling Contractors."

Benjamin F. Fairless Award to **Leo F. Reinartz,** "For distinguished achievement in production and in technology. For unerring guidance to the beginning metallurgist, invaluable advice to the operator. For inspiration to both."

Robert H. Richards Award to **Antoine Marc Gaudin,** "For his contributions as a scientist, educator, and author; specifically for his leadership and direction in the development of leaching and recovery techniques for low-grade uranium ores."

Robert W. Hunt Medal to **Arthur Tix.** Silver Medal for paper, *Production-Scale Vacuum Steel Degassing.* JOURNAL OF METALS, April 1956.

Rossiter W. Raymond Award to **Mohamed Mortada,** for paper *A Practical Method for Treating Oilfield Interference in Water-drive Reservoirs.* JOURNAL OF PETROLEUM TECHNOLOGY, December 1955.

Mathewson Medal to **Paul Gordon,** for paper, *Microcalorimetric Investigation of Recrystallization of Copper.* JOURNAL OF METALS, September 1955.

J. E. Johnson, Jr. Award to **R. W. Sundquist,** "For his ability as a Blast Furnace Operator in assisting in the development of new techniques and his contributions to the Blast Furnace Literature."

D. C. Jackling Award to **Joseph L. Gillson.**

Extractive Metallurgy Div. Award to **R. C. Bell, G. H. Turner,** and **E. Peters,** for paper *Fuming of Zinc from Lead Blast Furnace, Slag—A Thermodynamic Study.* JOURNAL OF METALS, March 1955.

Robert Peele Award to **Louis A. Panek,** for paper, *Analysis of Roof Bolting Systems Based on Model Studies.* MINING ENGINEERING, October 1955.

AIME Names 1957 Legion of Honor

Each year at the Annual Banquet in February, those members of the AIME who have continuously maintained their membership for 50 years are given special recognition. They are seated at the head table as guests of the Institute and are added to the membership of the Legion of Honor. Those who will achieve this status in 1957 are as follows:

Anderson, Alexander
Fullerton, Calif.
Brown, John T., Jr.
Pittsburgh, Pa.
Cook, Frederick S.
Mill Valley, Calif.
Dunster, C. B.
San Rafael, Calif.
Fisher, Norman H.
Montreal, Canada
Harrington, George B.
Chicago, Ill.
Hayward, Carle R.
Cambridge, Mass.
Knight, Cyril W.
Port Credit, Canada
Kruttschnitt, Julius
Brisbane, Australia
Lindberg, Carl O.
New York, N. Y.
Marshall, S. M.
Palo Alto, Calif.
McKechnie, B. E.
Carlsbad, N. M.
Rhodes, Fred N.
Takapuna, N. Z.
Ross, James G.
Thetford Mines, Can.
Sears, Stanley C.
Washington, D. C.
Shaw, S. F.
San Antonio, Texas
Weekes, Frederic R.
New York, N. Y.
Welhaven, Alf
Los Altos, Calif.
Faneuf, S. C.
Berkeley, Calif.

Panamerican Meeting Of Engineers in Mexico

The Fourth Convention of the Union Panamericana de Asociaciones de Ingenieros held in Mexico City, October 8-12, drew a registration of nearly 200 engineers from 16 countries in North, South and Central America, as well as in the West Indies.

Meetings were held in the Communications and Public Works Building during the week, to draft the various resolutions which were offered to the Convention on the final day.

Delegates were welcomed by the President of Mexico, His Excellency don Adolfo Ruiz Cortines, and James M. Todd, Chairman of the EJC delegation, who spoke on behalf of UPADI president, Luis Giannattasio. Speaking for the North American engineering societies was Gail Hathaway, chairman, EJC Committee on International Relations.

The Convention, which considered 33 resolutions, urged that formal sessions on engineering education be held simultaneously with future conventions of UPADI, that scholarships be made available to engineers to study in other countries, and that technical institutes be established for technicians and foremen. It suggested the compilation of a dictionary of common engineering terms, and noted the need for creating national standards organizations. The U. S. and Canadian delegations were invited to secure the adoption of the metric system in their countries. Lastly, all were asked to study means of increasing productivity.

The delegates once again chose Montevideo, Uruguay as UPADI headquarters and also decided to hold the 1958 convention in Montreal, Canada.

Officers of the convention were: President, Miguel A. Mantilla of Mexico; Vice Presidents, Adolph Ackerman, U. S.; Saturnino de Brito, Jr., Brazil, and Harry Tippenhauer, Haiti.

Dues Bills in Mail

Pursuant to Article II, Section 2, of the bylaws of the AIME, notice is hereby given that dues for the year 1957 are payable January 1, 1957, as follows: Members and Associate Members, $20; Junior Members for the first six years of Junior Membership, $12, and thereafter, $17; Student Associates (including an annual subscription to a monthly journal), $4.50.

Dues bills were mailed during the middle of November. Prompt payment will assure uninterrupted receipt of the publications desired in 1957. If, for any reason, a bill is not received within a reasonable time, headquarters should be notified.

Mining Branch Program
Beneficiation

(Continued from page 1229)

Solids-Fluids Separation

W. S. Springer, Session Chairman: *The Rotobelt Filter—A New Tool in Minerals Benefication*, by C. F. Cornell, R. C. Emmett, and D. A. Dahlstrom; *Recovery of Uranium from Uraniferous Lignites*, by Edward S. Porter and Henry G. Petrow; *Some New Solvent Extraction Processes for Use in the Hydrometallurgical Treatment of Uranium, Thorium, and Vanadium Ores*, by K. B. Brown, D. J. Crouse, and C. F. Coleman; *Solvent Extraction of Uranium at Shiprock, N. M. Plant of Kerr-McGee Oil Industries*, by Wayne C. Hazen.

Operating Control and Materials Handling

Neil Plummer and N. J. Sather, Session Chairmen: *Automatic Controls and Milling Operations*, by H. E. Uhland and W. Barbarowicz; *A New Theory of Ore Sampling*, by Pierre M. Gy; *Tailings Disposal at the Silver Bell Concentrator*, by Russell Salter and Wayne H. Tuttle; *General Description of Tailings Disposal at Braden Copper Co., Teniente Mine, Sewell, Chile*, by R. W. Jigins.

Concentration

F. J. Windolph and W. C. Mason, Co-Chairmen: *The Relative Effectiveness of Sodium Silicate of Different Soda-to-Silica Ratios as Gangue Depressants in Non-Metallic Flotation*, by C. L. Sollenberger and R. B. Greenwalt; *Leach-Precipitation-Flotation of Semi-Oxidized Copper Ore from the Ray Mines Division, Ray, Ariz.*, by A. W. Last, Lucien Eaton, Jr., and J. L. Stevens; *Separan 2610*[a]*, A Flocculant for the Mining Industry*, by Merrill F. McCarty and Robert S. Olson; *Milling Practice at Lavendar Pit Concentrator*, by Henry Martin.

Industrial Minerals

Chemical Raw Materials

G. D. Emigh and R. E. Shrode, Co-Chairmen: *Euxenite—A New Chemical Raw Material* by Warren A. Wagner; *Barite in New Mexico*, by Eugene Callaghan; *Lithium*, by P. E. Landolt; *Titanium and Zirconium*, by K. C. Li; *Problems of Mining and Exploration of Salt*, by C. H. Jacoby.

Ceramic Raw Materials

Pauline Moyd and Frank R. Hunter, Co-Chairmen: *The Use of Alumina In Glass*, by E. W. Summers; *Flotation of Feldspar*, by Carroll Rogers, Jr.; *Dry Processing of Feldspathic Pegmatite*, by H. B. DuBois; *Nepheline*, by H. R. Deeth and H. Maidment; *Aplite*, by V. V. Kelsey; *Other Source Materials*, by R. W. Hopkins.

(Continued on page 1234)

Around the Sections

• The **Reno Sub-Section** held their regular luncheon meeting in the Nevada Room of the Mapes Hotel, Friday, October 12. Robert C. Horton, mining engineer of the Nevada Bureau of Mines, addressed the group of thirty-six engineers and guests, describing the prospects, developments and active mining operations in the state. His discussion was prefaced with the financial history of mining in the state, with its high periods of the Comstock era, the '20s, noting that none of these periods approach the value of mining in the state at this time. It was brought out that Nevada has one of the largest mercury mines in the country, and the highest grade of iron ore in the country. Almost every county in the state currently has mining development in progress. These vary from two to three man leases, to large scale mine development programs directed by some of the larger companies in the country. Mr. Horton led the lively discussion session which followed.

1957 Officers Elected

At the November Board meeting, the officers of the AIME for 1957, and six other Directors named to take office for three-year terms in that year, were officially declared elected. They are as follows: President-Elect, A. B. Kinzel; Vice Presidents, E. C. Babson and Roger V. Pierce; Directors, Luther Campbell, T. C. Frick, F. W. Strandberg, A. W. Thornton, J. S. Vanick, and Lamar Weaver.

• The **Adirondack Section** held their Annual Meeting in Syracuse on November 3. Arrangements for the traditional weekend were handled by Jack Burns. Highlighting the get-together was the Syracuse vs. Penn State football game on Saturday followed by a cocktail party and dinner. Members and guests had comfortable accommodations at the King George Motel replete with air conditioning, television, and a heated swimming pool, as well as conference rooms.

• The **Black Hills Section** met at the Red Barn in Rapid City, S. D., on October 17. Chairman William C. Campbell introduced guest speaker AIME President Grover J. Holt who discussed the Lake Superior Mining District. The 54 members who attended also saw a homestake movie, *Deep Gold*. Officers elected for 1957 are: Chairman, Alexander E. McHugh; Vice Chairmen, Edward H. Stevens and Rex Tario; Secretary-Treasurer, Ted M. Rizzi. The next meeting of the section will be held January 10, 1957 at the Rod & Gun Club Building, Lead, S. D. After the routine business meeting a Hycon Aerial Survey film will be shown, entitled *Portrait of the Earth*.

• The **Lehigh Valley Section** held their Fall Dinner Meeting on September 28 at the Hotel Bethlehem, Bethlehem, Pa. Featured speaker was John T. Sherman, assistant director for domestic procurement, Div. of Raw Materials, AEC. He illustrated his talk on mining, milling, and peacetime uses of uranium, with a color film.

A new 16 mm, sound, color film on artificial respiration is available. The 12 mm movie, *No Time To Spare*, **emphasizes the importance of immediate application of respiration in different types of accidents where asphyxiation results, such as electric shock, carbon monoxide poisoning, and near drowning. The techniques of artificial respiration are demonstrated by non-professionals in typical cases. Technical advisors include Warren Clayson, of the Canadian Red Cross and M. G. Griffith, professor at the University of Toronto. The movie, produced by Chetwynd Films Ltd., may be purchased for $125 from** *International Film Bureau Inc., 57 E. Jackson Blvd., Chicago.* **Price includes educational and TV rights.**

Blasting Symposium

The University of Minnesota campus was the scene of the Sixth Annual Drilling and Blasting Symposium held October 11-13, under the sponsorship of the School of Mines and Metallurgy and the Center for Continuation Study, in cooperation with the Minnesota Section of AIME. The three-day conference was attended by some 175 operating engineers and men in the fields of development and research, who exchanged ideas on the problems encountered in drilling and blasting.

Rotary and percussive drilling were the principal topics of the opening session, at which Ralph Simon described the results of research on percussive drilling at Battelle Memorial Institute. Charles Fairhurst of the University of Minnesota covered the principles of both types of drilling, and presented the advantages of a new rotary-percussive technique in rock drilling.

Later sessions were given over to explosives, blasting and blast effects, and the recording and interpretation of seismic wave phenomena, after which the operating experts were given a chance to elaborate on their practical experiences.

Papers presented at the sessions will be published in the *Proceedings of the Symposium*, to be available within the next three months. Send orders to: Center for Continuation Study at the University of Minnesota, Minneapolis.

All-Institute Session During Annual Meeting

For the first time, an All-Institute Meeting is scheduled during the Annual Meeting in New Orleans. A prominent speaker will address the meeting at 2 pm on Tuesday, February 26th, in the International Room at the Roosevelt Hotel, New Orleans. Following that, at 4 pm, the Annual Business Meeting of the AIME will be held in the same place. At a recent meeting of the AIME Board of Directors, the fourth Tuesday of February was approved for this latter meeting instead of the third Tuesday as is indicated in the AIME bylaws.

ECPD Representatives

AIME representatives on the Engineers' Council for Professional Development for the year which began Oct. 26, 1956, have been named by the Board as follows: Committees: Guidance, F. T. Sisco and J. R. Van Pelt; Education and Accreditation, John C. Calhoun, Jr., and Lysle E. Shaffer; Student Development, Curtis L. Wilson; Recognition, R. E. Kirk; Training, Marion Semchysen; Ethics, Clyde Williams; Information, R. A. Beals. The three representatives on the Council are: R. F. Baker, Oct. '58; John Fox, Oct. '57; and J. P. Nielsen, Oct. '59.

1957 Publications Policies Established

Pursuant to Article X of the bylaws of the AIME, the following information is hereby given as to the "conditions, prices, and terms under which the various classes of members, and Student Associates, severally, shall be privileged to obtain publications of the Institute during the ensuing year."

Publications authorized for issue in 1957 include the following: MINING ENGINEERING, published monthly, containing material, including technical papers, of interest to those engaged in exploration, mining geology and geophysics, metal, nonmetallic, and coal mining and beneficiation; and fuel technology. The JOURNAL OF METALS, published monthly, containing material, including technical papers, of interest to those engaged in nonferrous smelting and refining, iron and steel production, and physical metallurgy. The JOURNAL OF PETROLEUM TECHNOLOGY, published monthly in Dallas, containing material, including technical papers, of interest to those in petroleum, natural gas drilling and production.

Annual subscriptions to any one of the above journals will be provided all members in good standing without further charge, a subscription credit of \$6 for members and \$4 for Student Associates being included in the dues paid. (A member ceases to be in good standing if current dues are not paid by April 1). If more than one of the monthly journals is requested, \$4 extra will be charged for an annual subscription, or 75¢ for single copies of regular issues, and \$1.50 for special issues. The nonmember subscription price for each journal is \$8 in the Americas and U. S. possessions; foreign, \$10; for single issues 75¢ in the Americas; \$1.00 foreign for regular issues; \$1.50 for special issues. Student Associates will be entitled to the same privileges for all publications as members. AIME members subscribing to more than one of each of the three monthly journals will be billed at the nonmember rate of \$8 per year, domestic; \$10 foreign for the extra subscription(s).

Three volumes of *"Transactions"* are authorized for 1957 publication, as follows: No. 205, Mining Branch; No. 206, Metals Branch; and No. 207, Petroleum Branch. Volumes 205 and 207 will be available to members at \$3.50 each for a first copy if paid for in advance with dues; otherwise at the nonmember rate of \$7 less 30 pct, nonmembers, \$7 in the United States; foreign, \$7.50. Volume 206 will be available to members at \$4.50 each for a first copy if paid for in advance with dues; otherwise at the nonmember rate of \$9 less 30 pct; nonmembers, \$9 in the United States, foreign, \$9.75.

Special volumes now planned for publication in 1957 include the following: *Open Hearth Proceedings*, Vol. 40, price to AIME members \$7; nonmembers \$10. *Blast Furnace, Coke Oven, and Raw Materials Proceedings*, Vol. 16, price to AIME members \$7; to nonmembers \$10. *Electric Furnace Steel Proceedings*, Vol. 14, price to AIME members \$7; to nonmembers \$10. *Statistics of Oil and Gas Development and Production*, Vol. 11, covering data for the year 1956, AIME members \$5; nonmembers \$10. *Porphyry Coppers*, 2d edition, by A. B. Parsons, price to be determined. *A New Look at the Nature of the Open-Hearth Process*, by B. M. Larsen, \$2 to AIME members, \$3 to nonmembers. *Physical Chemistry of Steelmaking—Deoxidation of Steel*, by C. H. Herty; price to be determined. Volume on Mineral Economics, price to be determined. 1957 AIME Directory.

If dues are paid subsequent to January 31, back issues of Institute publications will be supplied only if adequate stocks are on hand. A member is not entitled to receive a volume of *Transactions*, or a special volume, in lieu of a monthly journal, free of charge on membership. Members in arrears for dues are not entitled to special members' prices.

Rocky Mountain Members may choose an annual subscription to one of the monthly journals on request.

(Continued from page 1233)

Cement, Lime and Gypsum

Walter E. Trauffer and Carl Clausen, Co-Chairmen: *Highlights of Ideal Cement Co. Plant at Houston, Texas*, by T. B. Douglas; *Heavy Concrete for Atomic Energy Installations*, by Rudy Valore; *New Pressure Shaft Lime Kiln*, by Henry Erasmus and Hans Leuenberger; *Lime Stabilization of Soils for Road and Engineering Purposes*, by R. S. Boynton; *Underground Mining of Gypsum*, by F. C. Appleyard.

Mineral Aggregates and Dimension Stone

Current Economic Status of the Aggregate Industry, by LeRoy Otis; *A Rapid Method for Determining the Durability of Ledge Rock*, by Joseph E. Gray; *Effects of Manufactured Aggregate on the Pumpability of Sand-Cement Grout*, by James M. Polatty; *Special Problems in Granite Aggregates*, by D. Castleberry; *Experimental Determination of Stresses in Granite Formations*, by C. Mould.

Ultra-Lightweight Aggregates Symposium

Technology of Perlite, by Walter H. Taschek; *Technology of Vermiculite* by J. A. Kelley; *Technology of Expanded Shale*, by Thomas E. Shufflebarger, Jr.; *Technology of Pumicite or of Fly Ash*, (Author's name will appear in a later issue of MINING ENGINEERING.)

General Session

Arthur L. Hall and George F. Pettinos, Jr., Co-Chairmen: *Beneficiation of Refractory Minerals*, by Leo L. Gill and Earl Leatham; *Relationships of Product Quality to End Uses for Diatomaceous Earth*, by C. A. Frankenhoff; *Coated Abrasives—Their Manufacture and Use*, by L. A. Hansen; *The Use and Physical Character of Arkansas Novaculite*, by Dr. Norman F. Williams; *Production of Oil Well Fracturing Sand*, by A. D. Bryant; *Production of Monazite from Alluvial Concentrates*, by J. Hall Carpenter and Robert F. Griffith.

Coal

Mining and Geology

Basic Coal Composition and its Relationship to Preparation and Carbonization, by William Spackman, Jr., Andre H. Brisse, and W. F. Berry; *Mine Mechanization in Alabama*, by Henry J. Hager; *Continuous Mining in Pitching Seams*, by R. M. Von Storch.

Mining and Econom'cs

Coal for the Steam Plants of the Southeast, by Ralph E. Kirk; *Steam Pollution Legislation of Importance to the Coal Mining Industry*, by Henry F. Hebley and James R. Garvey; *Application of Industrial Engineering Principles to Bituminous Coal Mining*, by T. Carl Shelton, Jr.

(Continued on page 1239)

To the Members of the American Institute of Mining, Metallurgical, and Petroleum Engineers Inc.:

The regular Annual Business Meeting of the Institute will be held on Tuesday afternoon, the 26th day of February 1957 following the All-Institute session at the Annual Meeting of the Institute in New Orleans.

Since the Annual Business Meeting in 1957 will be held on the *fourth* Tuesday of February instead of the *third* Tuesday as the Bylaws direct, due notice is hereby given of this change.

Dated: Nov. 16, 1956
New York, N. Y.

By Order of the Board
Ernest Kirkendall, Secretary

(Continued from page 1237)

Materials Handling and Safety

Developments in Belt Conveyors for Mining, by William Hanson; *Progress in Safety Engineering at Alabama Coal Mines*, by Lawrence Henderson; *Mine Drainage Problems of the Pennsylvania Anthracite Industry and their Proposed Solution*, by Henry A. Dierks.

Dust Collection and Drying

Dust Collection With the Microdyne, by A. Lee Barrett; *Thermal Drying Costs for Fine Coal*, by Harry Washburn; *Combined Air Cleaning and Heat Drying of Coal*, by F. P. Calhoun.

Washing

The Convertol Process, by William L. McMorris and Andre H. Brisse; *Viscosity as a Factor in Dense-Medium Cleaning of Coal*, by H. F. Yancey; *Chance Cone Operation at Mathies Mine*, by D. H. Davis; *Coal Flotation with Liquid Hydrocarbons*, by S. C. Sun and D. R. Mitchell.

Mining

Shaft Sinking

H. B. Spencer, Chairman: *Long Raises at Homestake*, by C. N. Kravig; *Large Diameter Mine Shafts by Rotary Drilling*, by T. N. Williamson and Victor Zeni; *Circular Concrete Shafts*, by T. M. Berry.

Southeastern U. S. Mining

Arnold A. Gustafson, Chairman: *Operation of All-Belt Haulage System at International Salt Co. Mine, Avery Island, La.*, by Erland Johnson; *Magnet Cove Barium Corp. Arkansas Operation*, by C. F. Talbot; *Practical Aspects of Jet Drilling of Nepheline Syenite*, by T. M. Howell; *Mining Methods in the East Texas Iron Ore Deposits*, by Vincent F. Malone; *Haulage Improvements in Tennessee Coal & Iron Mines*, by L. H. Johnson, other staff members.

Canadian Mining

J. L. Ramsdell, Chairman: *Caland Ore Co., Ltd. Dredging Operation at Steep Rock Lake*, by Phillip D. Pearson; *Accident Prevention in Canadian Metal Mines*, by Neil H. George; *Back Filling at Noranda*, by Francis E. Patton; *Mine Hoisting in Canada*, by Campbell M. Barrett.

All MGG Session I

J. K. Richardson, Chairman: *Uranium Utilization in Retrospect and Prospect*, by R. L. Doan; *The New Crestmore Mine*, by R. H. Wightman, P. B. Nalle, and C. D. Chandler.

All MGG Session II

Roy P. Full, Chairman: The Jackling Lecture; *Recent Developments in Geochemical Exploration*, by T. S. Lovering.

Geology

General Session

Ernest L. Ohle, Chairman: *Administration of Geologic Personnel*,

(Continued on page 1240)

(Continued from page 1239)
by Arthur E. Granger; *The Ore Knob Copper Deposit, Ashe County, N. C.*, by Philip Eckman; *Fracture Controls of Uranium Ore Deposits, Grants District, N. M.*, by Thomas W. Mitcham; *Marcona Iron Deposits of Peru*, by Weston Bourret; *Slump Breccia Ores in Southeast Missouri*, by Frank Snyder and Jim Odell.

Symposium

Criteria for Recognition of Syngenetic Sedimentary Ore Deposits, Thomas N. Walthier and Alan M. Bateman, Co-Chairmen: *Geology of Taconite* by Josiah Royce; *On the Origin of Sedimentary Manganese*, by Charles F. Park, Jr.; *Isotopes and the Origin of Sedimentary Ore Deposits*, by Lawrence J. Kulp; *Copper Mineralization in the White Pine Deposit, Michigan*, by John R. Rand; *A Proposed Placer Origin for Blind River Uranium Ores*, by David S. Robertson and N. C. Steenland; *Patterns of Ores in Sediments and Sedimentary Rocks*, by George W. Bain.

Latin American Geology

Willard C. Lacy and Georges Ordonez, Co-Chairmen: (Titles to be announced in a later issue of MINING ENGINEERING.)

Geophysics

General Session I

Frank T. Clifton and George V. Keller, Co-Chairmen: *The New Airborne Gravity Method*, by Hans Lundberg; *Subsurface Investigations of a Plant Site, Missouri*, by LeRoy Scharon, Robert Uhley, and Tsui Meidav; *The Case History of the Juniper Prospect*, by R. A. Barker and S. H. Ward; *Gravity Prospecting for Chromite Deposits in Camaguey Province, Cuba*, by Willard E. Davis, W. H. Jackson, and D. P. Richter; *Magnetic and Electromagnetic Investigations in Paska Township, District of Thunder Bay, Ont.*, by Hal Fleming.

General Session II

George R. Rogers and S. H. Ward, Co-Chairmen: *Geophysical Studies in Northern Minnesota*, by Gorden D. Bath; *Continuous Pulse-Transient Logging in Mineralized Bore Holes*, by George V. Keller; *The Practical Application of Diamond Drill Hole Resistivity Surveys*, by D. J. Salt; *Varian Nuclear Precession Magnetometer—Its Theory and Application*, by Kenneth E. Hunter; *The Effects of Multiple Scattering On the Energy Spectrum From a Buried Gamma Ray Source*, by Robert J. Uffen and Norman B. Massey.

Mineral Economics

General Session

Future Markets and Prices by George Cleaver; *Atomic Power for Remote Mines*, by J. E. Crawford; *Economics of Handling Solids via Pipeline*, by R. K. Bond; *Changing Coal Industry*, by Hubert E. Risser.

Mineral Industries in Latin America

Raw Materials Base for South American Industrialization, by Adam Stricker; *Role of U. S. Government in South American Mineral Development*, by Sumner Anderson; *U. S. Experience in Bolivia*, by L. C. Raymond; *Latin America and U. S. Mineral Industry*, by John Ridge.

Panel Discussion

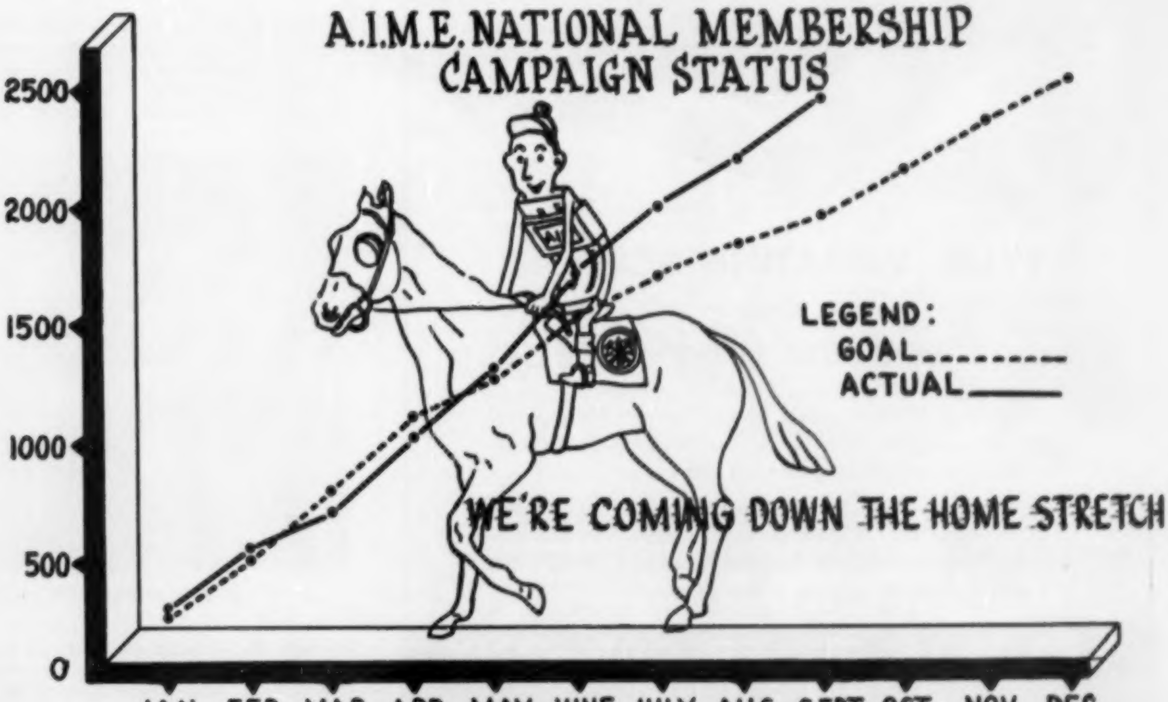
Economics of Research in the Mineral Industries: Panelists—Fayette Brown, Jr., E. W. Crabtree; Dean Calhoun, M. E. Volin.

Mineral Industry Education

As of November 1st, the MIED Annual Meeting Program for 1957 had not yet been finalized. MIED is planning two sessions on its technical program to be held on Sunday, February 24th at 2:30 and 8:00 pm.

Howe Lecturer Named

Maxwell Gensamer has been selected to be the Howe Lecturer in 1958, at the Annual Meeting in New York, 1958. Dr. Gensamer is Prof. of Metallurgical Engineering, School of Mines, Columbia University, N. Y.



PERSONALS

Emil J. Bonkoff is now associated with **Larry F. Labow** in Toronto as consulting mining engineer. Mr. Bonkoff, who holds a degree in geology from McMaster University, was formerly with the Department of Trade and Commerce in Ottawa, where he worked on the development and promotion of trade in industrial minerals and products for Canadian and foreign industries.

Arnold H. Miller, consulting engineer, is on professional visits to Spain, France, and Africa.

E. J. BONKOFF

John W. Hanley has been promoted to chief metallurgical engineer, Cerro de Pasco Corp., with offices in New York. A staff member since 1933, Mr. Hanley will have control over all the metallurgical functions and activities of the corporation.

David J. Crawford has become superintendent of mines, International Talc Co., Gouverneur, N. Y. He was formerly mining research engineer, Lehigh Navigation Coal Co., Lansford, Pa.

Gilbert C. Monture has been elected vice president of Strategic Materials Corp. and its subsidiaries. A recognized expert in mineral economics, Dr. Monture was a member of the NATO Raw Materials Conference in 1953 and the U. N. Committee to Study World Resources of Iron Ore in 1953 and 1954.

G. C. MONTURE

H. Van Arkel has left The Hague where he was consulting engineer, to become technical secretary of the board of Chemische Fabriek Naarden, Huizen, Holland.

H. VAN ARKEL

Herbert H. Kellogg has been promoted from associate to full professor of mineral engineering at the School of Mines, Columbia University, New York.

GROVER HOLT

Grover J. Holt, President-Elect of AIME, was the principal speaker at the dedication of the Ores Research Building, Bureau of Mineral Research, Michigan College of Mining & Technology, on August 8th. His address, *Why Be Concerned About Research?* drew attention to the fruits of past research and the need for its continuation. During the ceremonies, Mr. Holt was awarded an honorary Doctor of Engineering degree, the accompanying citation of which praised his "pronounced accomplishments to the mining profession" and his embodiment of "the ideals of service to your fellow men and the industrial society in which we live."

Mario Spada has left French Morocco to become manager of the mineral dressing department, University of Nancy, in France.

Harold P. Greenwald, Regional Director of the Bureau of Mines' Region V, retired on October 31 after 45 years of distinguished federal service. He had been in charge of operations in the states east of the Mississippi River and in Minnesota and Iowa. Mr. Greenwald directed fundamental research in coal, synthetic liquid fuels, explosives, iron ore, manganese, and other minerals in the region, in addition to compiling and publishing production data and mineral statistics. Appointed in 1949, Mr. Greenwald previously served for 13 years as superintendent of the Central Experiment Station, Pittsburgh. He was active in developing and directing research projects designed to promote health and safety in the mineral industries. Born in Boston in 1893, he obtained his B.S. degree in electrical engineering at Carnegie Institute of Technology and joined the USBM, Pittsburgh, in 1914. His professional affiliations include AIME, and the Coal Mining Institute of America.

Sherwin F. Kelly has been traveling in Central America, where his company was conducting geophysical surveys for metallic ores and volcanic sulfur. While there, he formed Kelly, Safie & Cía Ltda., which operates its properties under the name Minera Occidental Ltda.

Andrew Jenike, material handling consultant, Salt Lake City, is directing a new research project in the flow of bulk solids at the Utah Engineering Experiment Station, University of Utah. The project, partially financed by an Engineering Foundation grant, is being sponsored by the AIME Minerals Beneficiation Div.

C. H. Aall has left Monsanto Chemical Co. to become director of research and development, The American Metal Co. Ltd., Carteret, N. J.

Arthur J. Richards of Jersey City has taken a job with the AEC in Las Vegas, Nev.

Mord Lewis is vice president, Anaconda Aluminum Co., New York. He had been assistant to the vice president, Anaconda Copper Mining Co.

Leslie S. Voltz has been appointed director of safety for the operations of the Consumers Co., a division of Union Chemical & Materials Corp., in Illinois and Wisconsin. He had previously been resident engineer for the company on the contract construction of a new gravel and sand processing plant.

J. C. O'Donnell has been appointed general manager of Shaft & Development Machines Inc. This firm manufactures the new Cryderman mucking machine and markets the *Whup d' Whup* train car loader. Mr. O'Donnell had previously been mine superintendent for Idaho Maryland Mining Co. and Tungsten Mining Corp., prior to joining his present firm.

D. M. BENNETT

David M. Bennett has gone to Iraq with the Harza Engineering Co. and will work in Baghdad.

Allen T. Cole has opened a consulting office in Lakeland, Florida for the exploration, evaluation, mining, and processing of industrial minerals. Dr. Cole was formerly director of atomic energy services for W. R. Grace & Co.

George H. Love, president of Pittsburgh Consolidation Coal Co., has been honored by the renaming of the company's Northern W. Va. properties as the "Loveridge Mine".

John S. Sumner has been named manager of McPhar Geophysics Inc., Minneapolis.

Roy A. Anderson is chief geologist, Pend Oreille Mines & Metals Co., Metaline Falls, Wash.

J. F. Turner has resigned from Uruwira Minerals Ltd., Tanganyika, and is now metallurgist, Chibuhina Mines Ltd., Northern Rhodesia.

Clinton P. Mott, Kennecott Copper Corp., was elected president of the Great Salt Lake chapter, American Institute of Industrial Engineers.

David Roberts is being schooled in guided missiles at Fort Bliss, Texas, on reassignment from the Pentagon.

J. M. Rose is mining engineer, Mt. Isa Mines Ltd., Queensland, Australia. He was formerly associated with International Nickel Co., Sudbury, Ontario, Canada.

Louis W. Cope is assistant mill superintendent, Buchans Mining Co. Ltd., Newfoundland, Canada.

William G. Fischer is junior mining engineer at International Minerals & Chemical Corp., Carlsbad, N. M.

MEMBERSHIP

Proposed for Membership Mining Branch, AIME

Total AIME membership on Oct. 31, 1956, was 25,844; in addition 2,940 Student Associates were enrolled.

ADMISSIONS COMMITTEE

R. B. Caples, Chairman; F. A. Ayer, Vice-Chairman; A. C. Brinker, R. H. Dickson, C. R. Dodson, R. B. Fulton, T. D. Jones, F. W. Hanson, Sidney Rolfe, F. T. Sisco, O. B. J. Fraser, F. W. McQuiston, Jr., A. R. Lytle, L. P. Warriner.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

Members

Luther H. Baumgardner, Salt Lake City
R. H. Beck, Grand Junction, Colo.
Meredith C. Brown, Whittier, Calif.
John W. Crandall, Milwaukee
Richard E. Cribbs, Knoxville, Tenn.
Paul P. Eagan, Salt Lake City
Salich Faisl, New York
Dennis D. Foley, Columbus, Ohio
Richard C. Forbes, Eagle Mountain, Calif.
C. P. Gough, Vallejo, Calif.
Willie Hayes, Birmingham, Ala.
Thomas C. Hecker, Iron River, Mich.
Walter C. Hellyer, Telluride, Colo.
W. C. L. Hemeon, Pittsburgh
Chris H. Lee, Nipton, Calif.
Rolf Lentz, White Pine, Mich.
J. A. McAllister, Tucson, Ariz.
Daniel J. Miller, Jr., Sparta, N. J.
H. D. Milne, Cedar City, Utah
Joseph L. Monarchi, Denver
Karl W. Mote, Mountain View, Calif.
Merle J. Reed, East St. Louis, Ill.
William J. Roberts, Grants, N. M.
J. R. Ruark, Grand Junction, Colo.
John R. Ryan, Wilmington, Del.
Uuno M. Sahinen, Butte, Mont.
Ernest C. Simkins, Birmingham Canyon, Utah
P. S. Stevens, Lakeland, Fla.
Harold Stienmier, Gilman, Colo.
R. H. Sturgess, Snyder, N. Y.
Walter E. Thomas, Pittsburgh
Paul A. Wasson, Minneapolis
William J. Weik, Cornwall, Pa.

Associates

Leslie J. Battey, Salt Lake City
Glenn H. Clapp, Butte, Mont.
William E. Cotter, Jr., Duluth
Thomas F. Creamer, Scarsdale, N. Y.
Emilio Garate y Valdes, Mexico City, Mexico
Glenn D. Gurney, Canton, Ohio
L. E. Harbison, Morenci, Ariz.
John A. Hill, Jasper, Ala.
John R. Humphrey, New York
Malcolm Jones, Arvada, Colo.
Donald K. Lourie, New Canaan, Conn.
E. M. Michels, Phoenix, Ariz.
Laurence T. Nelson, Santa Monica, Calif.
Mario L. Pena, Mexico City, Mexico
Robert R. Portmess, Salt Lake City
Jacques Richards, Grand Junction, Colo.
Paul R. Scheier, Clifton, Ariz.
Herman Van Houten, Franklin, Pa.

Juniors

Raymond C. Barkley, Hot Springs, S. D.
Peter M. Bell, Bauxite, Ark.
Donald J. Brown, Richland, Wash.
Duane T. Cable, Pittsburgh
W. Bruce Clymer, Glassboro, N. J.
J. D. Covert, Grants, N. M.
George P. Dix, Jr., Grand Junction, Colo.
N. Dresner, Toronto
Robert B. Haagensen, Whitehouse, N. J.
Earl L. Hastings, University, Ala.
Milton A. Melcher, Colville, Wash.
Hans Neeb, Bogota, Colombia
J. Neilson, The Hague, Holland
Ivan C. Orelup, Rapid City, S. D.
Charles B. Pope, Silver Bell, Ariz.
M. Damaso Rojas, Morococha, Peru
Bruno Sabels, Miami, Ariz.
Charles R. Sewell, Freeport, Texas
H. T. Urband, Norfolk, Va.
Peter Vardy, Reno, Nev.
John Bruce Wallace, Houston

CHANGE OF STATUS

Junior to Member

Henry P. Ehrlinger, III, Reno, Nev.
David W. Tittman, Hoyt Lakes, Minn.

REINSTATEMENT

Members

W. T. Mahood, Wilmington, Del.
John R. Moore, Anaconda, Mont.
Carlos Orti, Madrid, Spain
Donald H. Selchow, Fernandina Beach, Fla.
W. M. Woodward, Salt Lake City

Associate

Fred James Watson, Salt Lake City

Juniors

Howard F. Bartlett, Lima, Peru
C. Frederick Eben, Annville, Pa.

REINSTATEMENT—CHANGE OF STATUS

Associate to Member

Wilmer R. Peterson, Salt Lake City
Myron L. Sisson, Sunrise, Wyo.
Wm. Bourne Wood, Reno, Nev.

Junior to Member

B. Barraza-O, Barranquilla, Colombia
Randall E. Brown, Richland, Wash.
John M. Conrow, Salt Lake City
E. M. Kennedy, Lautoka, Fiji Islands
John W. King, Rapid City, S. D.
Arne A. Mattila, Washington, D. C.
Charles F. B. Price, Jr., Walnut Creek, Calif.

Student to Member

A. J. Cigallio, Wakefield, Mich.
Vernon Gras, Cedar City, Utah
Reuben D. Larsen, Ruth, Nev.
E. O. Monge, Lawrence, Kan.
Haydn H. Murray, Bloomington, Ind.
Almer J. Peterson, Jr., Pittsburgh
John G. Smith, Ruth, Nev.
Edgar M. Zuercher, York, Pa.

Junior to Associate

James H. Maguire, Grants, N. M.

Student to Associate

J. H. Mosher, Hibbing, Minn.

Necrology

Date Elected	Name	Date of Death
1926	W. B. Coleman	Sept. 30, 1956
1935	Frank Eichelberger	Oct. 21, 1956
1922	Donald B. Gillies	Sept. 29, 1956
1949	Malcolm Glen	Unknown
1938	J. Spencer Hollings	June 1956
1883	Arthur H. Keller Legion of Honor	Unknown
1929	F. H. Kihlstedt	Oct. 19, 1956
1901	V. F. Stanley Low Legion of Honor	Unknown
1919	H. E. Nyberg	Oct. 18, 1956
1914	J. C. Pickering	Oct. 10, 1956
1940	James H. Ratliff	Sept. 20, 1956
1955	Joseph H. Show	Unknown
1920	Clair Upthegrove	Oct. 11, 1956
1948	R. G. Whealton	Unknown
1903	William Wraith Legion of Honor	Oct. 10, 1956

OBITUARIES

Donald Burton Gillies
An Appreciation by
Walter Carroll

Donald B. Gillies (Member 1922) died at his home in Cleveland Heights, Ohio, on September 29. He was born Nov. 4, 1872 at Bruce Mines, Ont., where his father was a miner. The family later moved to the copper country of the northern peninsula of Michigan.

Mr. Gillies graduated from the Mining School (Michigan College of Mining & Technology), from which he received his degree of mining engineer in 1893. In college he excelled in scholastic work and was a crack football player and track man. As early as 1890 he made the 100 yd dash in 10 seconds. In football he played right half.

Upon graduation he started West, ending up in Butte, Mont. The depression of '93 was at its height, and jobs were few and hard to get. Though a graduate mining engineer, Mr. Gillies got a job "pushing a slag pot," at the Montana Ore Purchasing Co.'s smelter.

He was soon promoted to the drafting room where he did extensive work on concentrator design. But Mr. Gillies' preference did not lie in an office or at a drafting table. He entered the active smelter field, working on reverberatory furnace practice in the production of copper matte, which was followed by a convertor operation in the production of pig copper.

DONALD B. GILLIES

As the depression came to an end he took a position as assistant assayer at the Parrott Silver & Copper Co. About this time Senator W. A. Clark decided to reopen the old copper blast furnace property in the Lost River District, 90 miles by stage from Blackfoot, Idaho. Mr. Gillies was made mining engineer for the W. A. Clark properties in Butte, which embraced both the copper and silver areas.

Promotions followed in quick succession. From mining engineer he was promoted to chief engineer, then to general superintendent and finally to manager of all of the Clark properties. It was while he was serving as manager that the famous Mayflower Mine was discovered and purchased by Mr. Gillies for Senator Clark. The property produced many millions before it ran out.

The desire to see even more of the West still remained active and three years were spent in the gold-silver fields of Tonopah and Goldfield during its most exciting development, when the Montana Tonopah and the Tonopah Extension mines were developed into dividend payers.

Mr. Gillies then turned his interests to Mexico, where in the state of Chihuahua he was active in mine management, engineering practice and mine examinations. He accepted management of the Mexican properties of Corrigan, McKinney & Co. of Cleveland, at the solicitation of Price McKinney, then president. These comprised the Rio Tinto Copper Co., and Concheno Mining Co.

In 1910 the Madero Revolution began in Mexico, causing great uncertainty and anxiety among Americans there, and during the next six years, existence was constantly tinged with real uncertainty. Pancho Villa ruled the northern states; raids were frequent and costly. In spite of political upheavals the properties managed by Mr. Gillies operated during the entire period and shipped ore and bullion without interruption or loss, except many hours of sleep.

In 1918 when the revolution ended, Mr. McKinney induced Mr. Gillies to come to Cleveland as vice president in charge of Corrigan, McKinney mining operations in the U. S. and Mexico. In this capacity he developed the Wolfpit & Greasy Creek coal mines in Pike County, Ky. Mr. Gillies held this position until the deaths of Mr. McKinney and Mr. James W. Corrigan, when he was promoted to president in 1932.

Mr. Gillies remained president of Corrigan, McKinney until 1935, when the company merged with Republic Steel Corp., and he joined Republic as a vice president.

In 1937 he interested Republic in the possibilities of the Northern Adirondacks and revived the nearly dormant iron ore industry in that area. At his recommendation Republic secured and still operates properties at Port Henry and Lyon Mountain, N. Y. In the late 1940's he investigated and brought to Republic's attention the rich iron ore deposit in Liberia, which the company cooperated in developing. This is perhaps the world's purest ore deposit, yielding 68 to 69 pct Fe.

In the last 10 years of his life—a time when most men would have been comfortably retired—he continued to search the world for new ore deposits. His later travels took him to Mexico, Venezuela, Liberia, Peru and Ecuador. On his 75th birthday, he was relieved of the vice presidency of Republic, but continued as mining consultant.

Mr. Gillies, AIME president in 1939, was also a member of the Mining & Metallurgical Society of America, the Montana Society of Engineers, and a trustee and member of the Cleveland Engineering Society.

His civic contributions have also been noteworthy. He was appointed chairman of the River and Harbor Committee, Cleveland Chamber of Commerce. Under his leadership, Cleveland's harbor has been tremendously improved and made accessible to constantly larger vessels. Without this work, Cleveland's iron and steel industry could not have reached its present proportion.

Mr. Gillies was also a member of the Lake Superior Iron Ore Assn., serving as president from 1947 to 1951, and as Chairman of its Board of Directors until his death.

He was active in the Cleveland Community Fund since 1932, and served on its Board of Trustees. In 1952 and 1953 he was Treasurer of the local chapter of the American Red Cross.

In 1953 he delivered the convocation address at the Michigan College of Mining and Technology, his alma mater, which had awarded him the degree of Doctor of Engineering in 1931. He also received an L.L.D. from the Montana School of Mines in 1939.

At 83, Mr. Gillies was an active, valuable member of his company and the community. His contributions to both, as well as to the steel industry and the nation, are of enduring value. Old in years he was energetic, active and an inspiration to all those who were associated with him. Mr. Gillies accomplished in one long lifetime the equivalent of the normal accomplishments of several men. He was active in three fields—nonferrous metals, coal, and iron ore, and made noteworthy contributions in each field, but especially in the field of ferrous metals during the last 20 years.

He was a member of the Union, Kirtland, Chagrin Valley Hunt, Pep-

Memorial Resolution

DONALD B. GILLIES

WHEREAS, with the passing of Donald B. Gillies at his home in Cleveland Heights, Ohio on September 29, 1956, the Institute has lost one of its most active, loyal, enthusiastic, and honored members; and
WHEREAS, during his many years of membership he took an enthusiastic part in the activities of the Institute. He served as Director in 1940 and 1941 and President in 1939; and
WHEREAS, during his 63 years of professional work his interests were in many phases connected with the mining and extraction of various metals, ferrous and non-ferrous; and
WHEREAS, in his progress from a humble laborer to a leader in industry he was the epitome of a true American; and
WHEREAS, he showed deep interest in assisting young men to enter our profession and in recognizing their progress and achievements; therefore, be it
RESOLVED, That the American Institute of Mining, Metallurgical, and Petroleum Engineers records with deep sorrow the passing of one of America's most distinguished engineers, and one of the Institute's best friends and advocates; and be it further
RESOLVED, That this Resolution be included in the minutes of this meeting and that a copy be sent to Mrs. Gillies.

November 16, 1956 — **Board of Directors**

per Pike and Mid-Day Tavern Clubs of Cleveland and the Miscowaubik Club of Calumet, Mich. He is survived by his wife, the former Mary Lou Yancey of Rome, Georgia, two daughters and a son, ten grand-children and six great-grandchildren.

Mr. Gillies' friends were legion, including presidents of countries, leaders of industry, on down through the spectrum of life to the corner bootblack. He was widely known as a friend and leader, world-wide contributor to the science of industry, and a wise counselor capable and willing to share his knowledge.

Folke H. Kihlstedt

An Appreciation by
Pomeroy C. Merrill

Folke H. Kihlstedt (Member 1929), an international pioneer in our profession, died at his home in Beach Haven, East New Market, Md., on Oct. 19, 1956, at the age of 55, after a long illness. He was a consulting engineer for U. S. Steel's Oliver Iron Mining Div. at the time of his death. He leaves many friends all over the world who will miss his smiling aid and encouragement.

Prior to his work at Duluth, Mr. Kihlstedt in 1947 was in charge of Oliver's exploration in Venezuela, which led to the discovery of Cerro Bolivar iron deposits. He returned to New York in 1951 with the U. S. Steel's Orinoco Mining Co. until he transferred back to the Oliver Div.

A native of Sweden, Mr. Kihlstedt graduated from the Royal Institute of Technology in 1923 and later became assistant professor of geology there. In 1927 he joined the Swedish Diamond Drilling Co., and for nine years was engaged in geophysical work in Europe, the U. S. and Central Africa. He went to the Philippines in 1935 and became superintendent of the Philippine Iron Mines at Larap. The war caught Mr. Kihlstedt and detained him in Manila until 1945. He then came to New York with Rogers, Mayer & Ball before going to Venezuela.

In addition to the Institute, Mr. Kihlstedt was a member of the Mining and Metallurgical Society of America, CIM, Geological Society of Sweden, Assn. of Engineers, Sweden, and the Mining Club, N. Y. One of the proudest days in his career was the day he obtained his U. S. citizenship in 1951.

Mr. Kihlstedt is survived by his wife, Carla Kihlstedt, and his children, Ingrid and Tyko.

Folke H. Kihlstedt Scholarship

His friends who are so widely scattered and who want to show their appreciation of his friendship, aid, and work, have established a Folke H. Kihlstedt Scholarship Fund for mining engineers, with the kind cooperation of the Woman's Auxiliary of AIME. They are sending their contributions to Mrs. Reed W. Hyde, 84 Mountain Ave., Summit, N. J., payable to Educational Fund, WAAIME, for Folke H. Kihlstedt Scholarship Fund.

Clair Upthegrove (Member 1920) professor emeritus of metallurgical engineering at the University of Michigan, died at his home in Ann Arbor on October 11. He was born at Central Lake, Mich., on Nov. 30, 1884, and earned a degree in chemical engineering at the University of Michigan, joining the faculty in 1916. Before commencing his teaching career, he was employed as an assistant chemist at the Oliver Mining Co. of Ironwood, Mich., and metallurgist for the Detroit Copper & Brass Rolling Mills. During World War I, he was an ordnance officer in the Army. An outstanding teacher, Professor Upthegrove also made valuable contributions to the field of metallurgy of nonferrous metals and diffusion in metals. His articles appeared in many professional journals. In addition to the AIME, Prof. Upthegrove served in the ASM, ASTM, the American Foundrymen's Assn., and the Society for Promotion of Engineering Education.

Appraisals
Assayers
Chemists
Construction
Consulting
Designing

Professional Services

Space limited to AIME members or to companies that have at least one member on their staffs. One inch, $40 per year; half inch, $25 payable in advance.

Geophysicists
Drilling
Management
Metallurgical
Reports
Valuations

ADAIR, BAILEY & VAN HORN
Minerals Consultants
Geology Mining Ore Dressing
Box 221, Murphy, N. C.

SIDNEY S. ALDERMAN, JR.
Consulting Geologist
814 Newhouse Building
Salt Lake City 11, Utah
Telephone: ELgin 9-0976

JAMES A. BARR
Consulting Engineer
Mt. Pleasant, Tennessee

Philip J. Baukol Member AIME • ASCE Reg. Mechanical Engr.
DESIGNER OF PLANTS
• INDUSTRIAL • METALLURGICAL •
2131 University Ave Berkeley 4, Calif.

BEHRE DOLBEAR & COMPANY
Consulting Mining Engineers and Geologists
11 Broadway New York 4, N. Y.

BLANDFORD C. BURGESS
Registered Professional Engineer
Mining Consultant
Monticello, Georgia

COWIN & COMPANY, INC.
Mining Engineers and Contractors
Shaft & Slope Sinking • Mine Development
Mine Plant Construction
1-18th Street SW,
Birmingham, Ala. Phone 56-5566

RENE ENGEL
Consulting Geologist
1222 Blair Avenue
South Pasadena, California
Telephone: ALbany 0912

DAVID LeCOUNT EVANS
Consultant
Mining Geology Petroleum Geology
314 Brown Bldg. Wichita, Kansas
Tel.: AMherst 2-8954 or MUrray 3-6437

HOWARD M. FOWLER
MINING ENGINEER
P.Eng.: British Columbia & Alaska
408 Rogers Bldg. • Vancouver, B. C.
Telephone: Tatlow 0729
Aircraft — Scintillometer equipped

CARLTON D. HULIN
Mining Geology
7 Ardilla Road Orinda, California

GUY E. INGERSOLL
Registered Professional Engineer in Texas, Arizona and New Mexico
Mine Examinations and Geological Reports
5505 Timberwolf Drive El Paso, Texas

C. PHILIP JENNEY
Consulting Geologist
372 Lakeshore Hy. W.
Oakville Ontario
Victor 4-9413

PHILIP L. JONES
Consultant
Mineral Economics & Mineral Dressing
Heavy Media Specialist
405 Miners Bank Bldg. Joplin, Mo.
Tel. MAyfair 3-7161

RAPHAEL G. KAZMANN
Consulting Ground-Water Engineer
Stuttgart, Arkansas

C. P. KEEGEL
Mining and Metallurgical Engineer
Administration Appraisal
Specializing in Management and Consultation in Latin America
1721 So. 14th St., Las Vegas, Nevada
Telephone DUdley 4-6081

KELLOGG EXPLORATION COMPANY
Geologists-Geophysicists
Air, Ground Surveys and Interpretation
3301 No. Marengo, Altadena, Calif.
Sycamore 4-1973

KELLOGG KREBS
Mineral Dressing Consultant
564 Market St., San Francisco 4, Calif.

KIRK & COWIN
Consulting • Appraisals • Reports
1-18th Street SW,
Birmingham, Ala. Phone 56-5566

LEDOUX & COMPANY
Chemists Assayers Spectroscopists
SHIPPERS REPRESENTATIVES
Mine Examination Analyses
359 Alfred Ave. Teaneck, New Jersey

JOSEPH T. MATSON
CONSULTING MINING ENGINEER
Examinations—Appraisals Operations
P. O. Box 170 Santa Fe, New Mexico

CLAYTON T. McNEIL, E. M.
Consulting Mining Engineer
822 Bank of America Bldg.
Tel. GArfield 1-2948
SAN FRANCISCO 4, CALIFORNIA

ARNOLD H. MILLER
Consulting Engineer
Mine, Mill and Industrial Investigations
Improvement Design and Recommendations
Cable: "ALMIL" Tel. Cortlandt 7-0635
120 Broadway New York 5, N. Y.

O'DONNELL & SCHMIDT
Mining Consultants
165 Broadway Tel. BArclay 7-6960
New York 6, N. Y. Cables: EXAMIMINES

AMEDEE A. PEUGNET
CONSULTING MINING ENGINEER
Telephone MAIN 1-1431
705 Chestnut St. St. Louis 1, Mo.

LUCIUS PITKIN, INC.
Mineralogists
Assayers—Chemists—Spectroscopists
Shippers' Representatives
PITKIN BLDG., 47 FULTON ST., NEW YORK
Cable Address: Niktip

MILNOR ROBERTS *Consulting Mining Engineer*
The Pacific Northwest,
British Columbia and Alaska
4501 15th Ave., N.E. Seattle, Wash.

WILLIAM J. SHEDWICK, JR.
Mine and Geologic Reports
Mexico and Latin America
New Jersey License 2744-a
Reforma 20-302 Mexico 1, D.F.

CLOYD M. SMITH
Mining Engineer
Mine Examinations
Ventilation Surveys
Munsey Building Washington 4, D.C.

LEO H. TIMMINS, P. Eng.
MINING ENGINEER
Examinations - Reports
Financing of Prospects
Suite 700 1980 Sherbrooke, Montreal
Phone Glenview 2376

GODFREY B. WALKER
Metallurgical Consultant
Mineral Dressing & Extractive Metallurgy
Heavy Media a Specialty
27 Lockwood Drive Old Greenwich, Conn.

O. W. WALVOORD CO.
Mill-Design and Construction
401 High St. Denver 3, Colo.

CLIFFORD R. WILFLEY
Consulting Mining Engineer
2233 Grape St. Denver 7, Colorado

CLYDE H. WILSON
MINING ENGINEER AND GEOLOGIST
Registered Professional Engineer
GEOLOGICAL & GEOPHYSICAL SURVEYS
Mineral Deposits • Petroleum
Ground Water
366 South Fifth East, Salt Lake City 2, Utah

HARRY J. WOLF
Mining and Consulting Engineer
Examinations—Valuations—Management
One Park Place, New York 7, N. Y.
Cable: MINEWOLF Tel. Rector 2-5207

NEWELL G. ALFORD
Consulting Mining Engineer
Coal Property Prospecting, Development, Operation and Valuation
Oliver Building Pittsburgh 22, Pa.

ALLEN & GARCIA COMPANY
42 Years' Service to the Coal and Salt Industries as Consultants, Constructing Engineers and Managers
Authoritative Reports and Appraisals
332 S. MICHIGAN AVE., CHICAGO
120 WALL ST., NEW YORK CITY

BALL ASSOCIATES
Oil, Gas and Minerals Consultants
Douglas Ball Wendell W ertig
R. H. Fulton Alan M. Bieber
A. S. Wyner
Offices
1025 Vermont Ave. C. A. Johnson Bldg.
Washington, D. C. Denver, Colo
STerling 3-1929 ALpine 5-4878

B. B. R. DRILLING CO.
National Road West
St. Clairsville, Ohio
Diamond Core Drilling Contractors
Mineral Cores Guaranteed — Foundation Testing

CENTENNIAL DEVELOPMENT CO.
Consulting Mining Engineers and Contractors
Shaft Sinking — Tunnel Driving
Mine Development
Eureka, Utah — Phone 560

EAVENSON, AUCHMUTY & SUMMERS
MINING ENGINEERS
Mine Operation Consultants
Coal Property Valuations
2720 Koppers Bldg. Pittsburgh 19, Pa.

MAC AFEE and COMPANY
Consulting Engineers
MINE EXAMINATION & MILL DESIGN
Exploration • Construction
Operation & Management
CHEMICAL & METALLURGICAL LABORATORY
FOR EXACT QUANTITATIVE FLOWSHEET DATA
3105 Wilshire Boulevard • Los Angeles 5
DUnkirk 8-9674 • CABLE MACAFEE

FAIRCHILD AERIAL SURVEYS, INC.
Airborne Magnetometer & Gradiometer Surveys, Topographic Mapping, Aerial Photography, and Photographic Mosaics for Mining Exploration.
224 E. 11th St. Los Angeles — 30 Rockefeller Plaza New York

GRAFF ENGINEERING COMPANY
Mining Engineers and Surveyors
39 E. Campbell St. — Blairsville, Pa.

ABBOT A. HANKS, Inc.
ASSAYERS-CHEMISTS
Shippers Representatives
624 Sacramento Street
SAN FRANCISCO

JOHN D. HESS
Consulting Ground-Water Geologist
- Investigations
- Reports
- Geochemical Studies
- Electrical Logging
- Drainage Surveys
- Water Supply

Complete Physical-Chemical Laboratory Facilities
EL CENTRO, CALIFORNIA

HOPKINS EXPLORATION CONSULTANTS
607-320 Bay St. — Toronto 1
EM. 4-5642 — HU. 9-8375
Branches
Algoma Mills and Timmins, Ont.
Uranium City, Saskatchewan

DIAMOND CORE DRILLING BY CONTRACT
and world's largest manufacturer
Core and grout hole drilling in coal, metal, and non-metallic deposits, both surface and underground.
JOY MANUFACTURING CO.
Contract Core Drill Division
Michigan City, Indiana

KNOWLES ASSOCIATES
Chemical - Metallurgical - Mechanical
ENGINEERS
URANIUM ORE PROCESSING
ECONOMIC STUDIES - MILL DESIGN
19 RECTOR ST. — NEW YORK (6) N. Y.

LEGGETTE, BRASHEARS & GRAHAM
Consulting Ground-Water Geologists
Water Supply — Salt Water Problems
Dewatering — Investigations
Recharging — Reports
551 Fifth Avenue, New York 17, N. Y.

ROBERT S. MAYO
Civil Engineer — Lancaster, Pa.
Specializing in Concrete Lining of Tunnels, Haulageways and Shafts. Special Equipment for Subaqueous Construction.

E. J. LONGYEAR COMPANY
Geological and Mining Consultants
Mineral Exploration and Appraisals
Photogeology
Foshay Tower — Mpls. 2, Minn.
Graybar Bldg. — N. Y. 17, N. Y.
Colorado Bldg. — Denver 2, Colo.
Shoreham Bldg. — Wash. 5, D. C.

R. S. MC CLINTOCK DIAMOND DRILL CO.
Spokane, Wash. — Globe, Ariz.
Diamond Core Drill Contractors
Manufacturers of Diamond Bits and Drilling Accessories

JOHN F. MEISSNER ENGINEERS, INC.
Consulting Engineers
Conveyor Systems — Storage Methods
Crushing Plants — Ship Loading Docks
Materials Handling and Processing Plants
308 W. Washington St. Chicago 6, Ill.

JOHN D. MORGAN, JR., E. M., PH. D.
Consultant
Business and Defense Problems in Metals, Minerals, and Fuels
724 14th St., N.W., Washington 5, D.C.
ME 8-1681

J. B. MORROW
COAL CONSULTANT
Oliver Bldg. — Pittsburgh, Pa.

DIAMOND CORE DRILLING CONTRACTORS
Testing Mineral Deposits
Foundation Borings
MOTT CORE DRILLING CO.
Huntington, W. Va.

Frank M. Murphy & Associates, Inc.
DESIGN & CONSTRUCTION ENGINEERS
Specializing in Material Handling Mining Mineral Beneficiation & Chemical Processing
Wesley M. Houston — John C. Yost
J. D. Raulerson, Jr.
Box 271 — Bartow, Fla.

PENNSYLVANIA DRILLING COMPANY
PITTSBURGH 20, PA.
DRILLING CONTRACTORS and MANUFACTURERS
We prospect coal and mineral land anywhere in North and South America.
Core borings for foundation testing; dams, bridges, buildings, etc.

PRODUCTION AND MANAGEMENT SPECIALIST
ROGER V. PIERCE
Underground Mining Methods, Cost Cutting Surveys—Production Analysis—Mine Mechanization—Mine Management.
808 Newhouse Bldg. EMPIRE 3-5373
Salt Lake City 4, Utah

SHENON AND FULL
Consulting Mining Geologists
700 Newhouse Building
Salt Lake City 11, Utah
Philip J. Shenon — Roy P. Full

M. G. SMERCHANSKI
Consulting Mining Geologist
Registered Professional Engineer
Examinations, Geological Surveys & Development
411 Childs Bldg. Winnipeg, Manitoba
Phone: 926323

SPRAGUE & HENWOOD, Inc.
SCRANTON 2, PA.
Diamond Drill Contractors and Manufacturers
Core borings for testing mineral deposits in any part of the world.

STILL & STILL
Consulting Mining Engineers and Geologists
24 Union Block — Phone 658
P.O. Box 1512
Prescott, Arizona

H. L. TALBOT
Consulting Metallurgical Engineer
Extraction and Refining of Base Metals
Specializing in Cobalt and Copper
Room 330, 84 State Street
Boston 9, Mass.

PAUL WEIR COMPANY
Mining Engineers and Geologists
Consultants and Managers
Design and Construction
20 No. Wacker Drive — Chicago 6, Ill.

J. W. WOOMER & ASSOCIATES
Consulting Mining Engineers
Modern Mining Systems and Designs
Foreign and Domestic Mining Reports
Henry W. Oliver Bldg., Pittsburgh, Pa.

WORLD MINING CONSULTANTS, INC.
Consulting Mining Engineers and Geologists
220 Broadway, New York 38, N. Y.
Worth 2-2934

Coming Events

Dec. 5-7, AIME, Electric Furnace Steel Conference, Hotel Morrison, Chicago.

Jan. 9-11, 1957, Fifth Annual Seminar on Spectroscopy and Southern Assn. of Spectrographers' Annual Meeting, University of Florida, Gainesville, Fla.

Jan. 17-18, EJC, annual meeting, Statler Hotel, New York.

Feb. 7-9, Colorado Mining Assn., Denver.

Feb. 24-28, AIME Annual Meeting. Roosevelt and Jung Hotels, New Orleans.

Mar. 10-16, EJC Second Nuclear Engineering and Science Congress, Convention Hall, Philadelphia.

Apr. 1-4, Amer. Assn. of Petroleum Geologists, annual meeting, Kiel Auditorium, St. Louis.

Apr. 5, 6, AIME Pacific Southwest Mineral Industry Conference. Sponsored by Nevada Section, cosponsored by San Francisco and Southern California Sections. Reno.

Apr. 8-10, AIME, National Open Hearth Steel and Blast Furnace, Coke Oven, and Raw Materials Conferences, William Penn Hotel, Pittsburgh.

Apr. 11-13, AIME, Pacific Northwest Regional Conference, Multnomah Hotel, Portland, Ore.

Apr. 22-24, Second Annual Symposium on Rock Mechanics, Colorado School of Mines, Golden, Colo.

Apr. 22-24, CIM, annual meeting, Ottawa, Ont., Canada.

May 13-16, Coal Convention and Exposition of the American Mining Congress, City Auditorium, Cleveland.

Sept. 8-Oct. 9, Commonwealth Mining and Metallurgical Congress, British Columbia to Nova Scotia, Canada.

Sept. 9-12, American Mining Congress, annual convention, Utah and Newhouse Hotels, Salt Lake City.

Sept. 18-21, International Mineral Dressing Congress, Royal Inst. of Technology, Stockholm, Sweden.

Oct. 6-9, AIME, Petroleum Branch, Dallas.

Oct. 9-11, ASME-AIME Coal Div., Joint Solid Fuels Conference, Chateau Frontenac, Quebec.

Oct. 15-18, Northeastern Mining Branch Conference, Hillsboro Hotel, Tampa, Fla.

Oct. 30-Nov. 1, AIME, Rocky Mountain Minerals Conference, Denver.

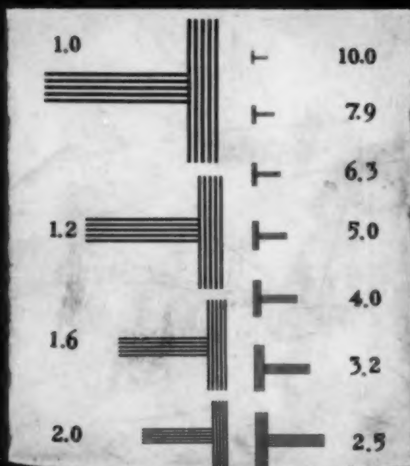
Nov. 4-6, AIME, IMD Fall Meeting, Morrison Hotel, Chicago.

Nov. 11-14, Society of Exploration Geophysicists, 27th Annual Meeting, Statler-Hilton Hotel, Dallas.

Advertisers Index

* Previous Issues

RESOLUTION CHART



100 MILLIMETERS

INSTRUCTIONS Resolution is expressed in terms of the lines per millimeter recorded by a particular film under specified conditions. Numerals in chart indicate the number of lines per millimeter in adjacent "T-shaped" groupings.

In microfilming, it is necessary to determine the reduction ratio and multiply the number of lines in the chart by this value to find the number of lines recorded by the film. As an aid in determining the reduction ratio, the line above is 100 millimeters in length. Measuring this line in the film image and dividing the length into 100 gives the reduction ratio. Example: the line is 20 mm. long in the film image, and 100/20 = 5.

Examine "T-shaped" line groupings in the film with microscope, and note the number adjacent to finest lines recorded sharply and distinctly. Multiply this number by the reduction factor to obtain resolving power in lines per millimeter. Example: 7.9 group of lines is clearly recorded while lines in the 10.0 group are not distinctly separated. Reduction ratio is 5, and 7.9 x 5 = 39.5 lines per millimeter recorded satisfactorily. 10.0 x 5 = 50 lines per millimeter which are not recorded satisfactorily. Under the particular conditions, maximum resolution is between 39.5 and 50 lines per millimeter.

Resolution, as measured on the film, is a test of the entire photographic system, including lens, exposure, processing, and other factors. These rarely utilize maximum resolution of the film. Vibrations during exposure, lack of critical focus, and exposures yielding very dense negatives are to be avoided.